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# AMERICAN INSTITUTE OF MINING ENGINEERS

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CONTAINING PAPERS AND DISCUSSIONS PRESENTED AT
THE ST. LOUIS MEETING, OCTOBER, 1917,
AND AT THE NEW YORK MEETING,
FEBRUARY, 1918.

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# PREFACE

This volume contains papers on Iron and Steel, Ore Deposits, and Miscellaneous Subjects, presented at the St. Louis Meeting in October, 1917, but omitted from Volume LVII for lack of space. These papers were published in *Bulletins* 123 to 130, inclusive; this volume of *Transactions* therefore replaces all *Bulletins* up to No. 130, October, 1917, inclusive.

To the above-mentioned St. Louis papers, have been added the papers on Iron and Steel, and the Metallography of Steel, presented at the New York Meeting, February, 1918, and printed in *Bulletins* 131 to 134, inclusive.

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## PAPERS

# PAPERS

# Comparative Tests of Hammer Drill Bits

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(St Louis Meeting, October, 1917)

## Introduction

MANY different shapes of drill bits are in use with hammer drills, but little definite information is available whereby to judge which one of these shapes is the best.

The following investigation was undertaken in order to determine what effect, if any, the shape of the cutting edge has on the cutting speed and wearing qualities of drill bits.

The bits tested were those in common use, namely, the 4-point or cross bit, the 6-point bit, the Z bit and the "Carr" bit. The rock in which the tests were made was the red granite from southeast Missouri. It is a coarse crystalline granite containing an unusually large amount of quartz and is extremely hard, and on account of its uniform texture is admirably adapted to this work. While tests in other rocks might show somewhat different results, nevertheless it is the opinion of the authors that the relative cutting quality inherent in the shape of the bits would be the same in all rocks. Other qualities, such as mudding freely, freedom from fitchering, etc., might make one bit more desirable than another in softer rocks.

## Description of Bits Tested

The 4-point bits were the usual shaped bits commonly made on the Leyner sharpener, with 14° taper on the wings and a 90° angle between cutting edges. Some tests were run at 85 lb. pressure with a 4-point bit made with a 5° taper on the wings; this bit was formed with the Carr bit dies.

The 6-point bits were the usual shaped bits made on the Leyner sharpener.

The Carr bits were made with a 5° taper and an angle of 100° between cutting edges.

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The Z bits were formed with the regular Z-bit dolly which makes an angle of about 60° in the center and an angle of about 45° on the outer cutting edges. The center is made slightly higher than the outer edges. The Carr bit dies were used in forming Z bits to give them a 5° taper. The Z-bit dolly was for solid steel, so it was necessary to drill out the center hole.

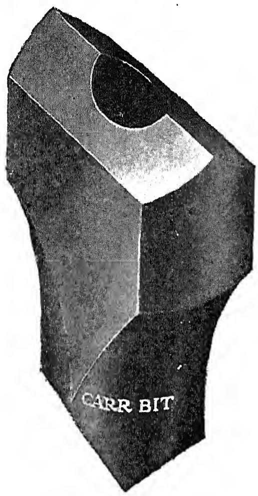


Fig. 1.

Several Z bits were tested at 85 lb. pressure, having been filed and hammered down to give an angle of about 100° on the center edge and 65° on the outer edges.

The holes in the center of the bits were the ordinary size with the 4-point and 6-point and Carr bits. The hole in the Carr bit was much larger than those used in the others. A hole of the same size as in the Carr bits was used in the Z bits.

## Manner of Conducting Tests

All tests were made with an Ingersoll-Rand "Jackhamer" drill. In the "down-hole" tests the drill was weighted with a 94-lb. weight, thus insuring a constant pressure on the bit.

Most of the tests were made in vertical "up-holes," great care being taken to see that the drill was vertical. The tests consisted of drilling into a large block of granite supported on stringers over a concrete-lined pit as shown in the photograph (Fig. 2). For this work the drill was mounted on an air feed, which, however, was not large enough to produce the required pressure on the bit, so that it was necessary to counterbalance the weight of the machine by a bucket loaded with 40 lb. of scrap iron and connected to the machine by a rope over an overhead pulley.

Every effort was made to insure uniform conditions and to keep the drill bit the only variable.

### *Air Pressure*

A uniform air pressure was obtained by having a "pop-off" valve on the air receiver, which was connected by a 1-in. hose to the drill. The air compressor was regulated to supply sufficient air to keep the pop-off valve open and the drill running at the same time, the gage pressure remaining constant. The air receiver was drained after every third steel, as it was found in former work that this factor greatly influenced the quality of the air supply.

### *Lubrication*

Uniform lubrication is most essential in work of this nature and was

insured by oiling the drill after each 1-min. test. The drill was taken apart and thoroughly cleaned and oiled after every third steel.

*Steel*

The steel used was all of the same kind, F J A B 7/8-in. hollow hexagon, and all comparative tests were run with steel of practically the same length. Lengths of 24, 48 and 62 in. were used.

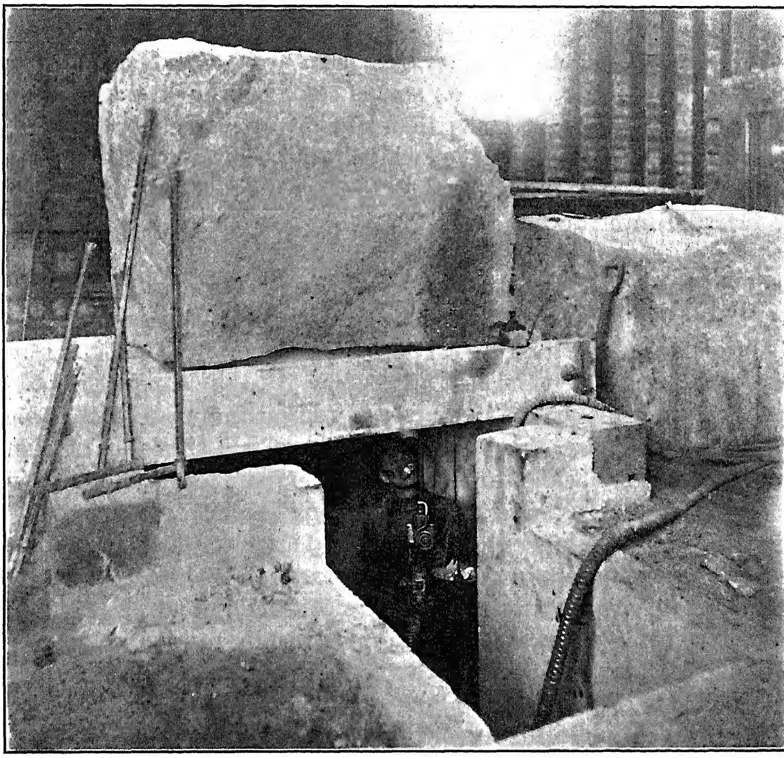

FIG. 2.—METHOD OF DRILLING "UP" HOLES.

*Sharpening and Tempering*

The greatest source of error in work of this kind is in the making and tempering of bits. It is impossible to make these conditions entirely uniform and many tests were made and the results discarded, because of non-uniformity in tempering. Any bit with a corner off, or which was chipped in any way (except the Z bit in a few cases), or which showed signs of being too hard or too soft, was rejected and resharpened until a perfect temper was obtained.

All bits were made and sharpened in a Leyner-Ingersoll 5A sharpener. The bits were heated, both for sharpening and tempering, in a gas muffle.

In sharpening and tempering the following rules were observed:

(*a*) Heat to the usual working heat, taking care to see that the bit is heated uniformly throughout, and making as few heats as possible.

(*b*) Heat only the end of the bit.

(*c*) Never submerge a bit entirely under water until cold.

(*d*) Keep shank end square.

(*e*) Never let scale form by too much air.

(*f*) When worked put in lime to anneal.

(*g*) Heat the bit up to at least 800° and be sure to leave it in the furnace long enough to heat to the center; then draw from the fire and place in some dark chamber (tile, etc.) so the color can be observed, until the steel has cooled to about 800°; then dip into Sentinel paste which has melting point of 775° C., and let white residue form on the bit; place in the furnace immediately. The white residue should melt the instant the bit is inserted into the fire again; if it does not melt at once, reheat and repeat the operation.

The Sentinel paste is a trade composition consisting of molecular mixtures of metallic salts which melt at predetermined temperatures. The finely ground salts are mixed with paraffin wax. Before using, it is heated gently so as just to melt the wax and reduce it to the consistency of thick paint. It can then be streaked or painted on the work in the cold; or, as in the present case, the work is covered with a thin film of paste by dipping in the paste while the work is at a temperature above the melting point of the salts. When the wax carrying the Sentinel powder burns off, it leaves behind a layer of white salt, and upon the desired temperature being attained, the salt fuses and disappears, or, on a reduction of temperature, the paste reappears as a white coating.

After the bit is put in the furnace and the white residue melts instantly, plunge the bit into cold water to about ¼ to ½ in.

All bits in these tests were plunged into water as described above and slowly moved about in the water until fairly cool; then the steel was left in water until cold.

## Experiments with Down Holes

The first series of tests was made in down holes. The purpose of these tests was to ascertain the effect of gage or diameter of bit on cutting speed. It was necessary to find this relation in order to compare different bits by reducing them to the same gage.

All tests were made in shallow holes, about 1 ft. in depth, so that the cuttings could be easily blown from the hole.

Down-hole tests were made only with the 4-point bit and at a pressure of 95 lb. In most cases, five 1-min. runs were made with a bit.

About 30 tests were made altogether. Table 1 shows the results of 12 of the most consistent of these tests.

TABLE 1.—*Results from Down Holes, 95 Lb. Pressure*

| No of Test | Gage at Start | Meas. of Hole | Dist. Per Min. | Gage at Start | Meas. of Hole | Dist Per Min. | Gage at Start | Meas. of Hole | Dist Per Min |
|---|---|---|---|---|---|---|---|---|---|
| | 1.600 | 1.80 | ....... | 1.750 | 4 90 | ...... | 1.765 | 1.35 | |
| 1 | 1 560 | 5.60 | 3.80 | 1.650 | 7.60 | 2.70 | 1.750 | 4 45 | 3.10 ] |
| 2 | 1.550 | 9 00 | 3.40 | 1.625 | 9.90 | 2.30 | 1.713 | 7.00 | 2.55 |
| 3 | 1.530 | 12.20 | 3.20 | 1.625 | 12.05 | 2.15 | 1.710 | 9.50 | 2.50 |
| 4 | 1.530 | 14.20 | 2.00 | 1.625 | 13.90 | 1.85 | 1.690 | 11.50 | 2.00 |
| 5 | 1.500 | 16.50 | 2.30 | 1.625 | 15 40 | 1 50 | 1.690 | 13.45 | 1.95 |
| 6 | ........ | ....... | ..... | ...... | .. .. | ....... | 1.690 | 15 25 | 1.80 |
| | 1.920 | 1.05 | ...... | 2.125 | 1 35 | ....... | 1.580 | 3.70 | |
| 1 | 1.875 | 3 70 | 2.65 | 2.125 | 3.45 | 2.10 | 1.560 | 7.55 | 3.85 |
| 2 | 1.845 | 6.00 | 2.30 | 2.095 | 5.30 | 1.85 | 1.530 | 10.75 | 3.20 |
| 3 | 1.815 | 7 85 | 1.85 | 2.060 | 7.15 | 1.85 | 1.530 | 13.85 | 3.10 |
| 4 | 1.815 | 9.60 | 1.75 | 2.060 | 8 80 | 1.65 | 1.520 | 16.90 | 3.05 |
| 5 | 1.815 | 11.35 | 1.75 | 2.060 | 10.35 | 1.55 | 1.500 | 19.75 | 2.85 |
| 6 | 1.800 | 12 85 | 1.50 | 2.047 | 11.70 | 1.35 | 1.500 | 22.15 | 2 40 |
| | 1 350 | 3.15 | ...... | 1.330 | 1.60 | ....... | 1.440 | 1.85 | |
| 1 | 1.300 | 8.75 | 5.60 | 1.315 | 6.50 | 4.90 | 1.420 | 6.35 | 4.50 |
| 2 | 1.300 | 13 15 | 4.40 | 1.305 | 10.60 | 4.10 | 1.400 | 10.10 | 3.75 |
| 3 | 1.300 | 17.45 | 4.30 | 1 300 | 14.80 | 4.20 | 1.370 | 13.10 | 3.00 |
| 4 | 1.300 | 21.35 | 3.90 | 1.245 | 18.10 | 3.30 | 1.370 | 16.15 | 3.05 |
| 5 | 1.240 | 24.90 | 3.55 | 1.240 | 21.25 | 3.15 | 1.370 | 18.95 | 2.80 |
| 6 | 1.230 | 28.30 | 3.40 | 1.235 | 23.80 | 2 55 | 1.360 | 21.40 | 2.45 |
| 7 | 1.225 | 31.60 | 3.30 | 1 230 | 26 90 | 3.10 | 1.350 | 23.75 | 2.25 |
| | New hole | | | | | | | | |
| 8 | 1.220 | 6.95 | 0 00 | 1 230 | 29 70 | 2.80 | 1.320 | 25.95 | 2.20 |
| 9 | 1.215 | 10.85 | 3.90 | | | | | | |
| | 1.470 | 1.85 | ...... | 1.590 | 2.10 | ....... | 1 560 | 6.10 | |
| 1 | 1.440 | 5 85 | 4.00 | 1.560 | 5.80 | 3.70 | 1.550 | 9.80 | 3.70 |
| 2 | 1.440 | 9.15 | 3.30 | 1.520 | 9.05 | 3.25 | 1.530 | 12.70 | 2.90 |
| 3 | 1.400 | 12.00 | 2.85 | 1.500 | 12.05 | 3.00 | 1.500 | 15.20 | 2 50 |
| 4 | 1 370 | 14.55 | 2.55 | 1.490 | 14.90 | 2.85 | 1.500 | 17.45 | 2.25 |
| 5 | 1.370 | 17.00 | 2.45 | 1.490 | 17.50 | 2.60 | 1.500 | 19.70 | 2.25 |

The accompanying curve (Fig. 3) was plotted assuming that the cutting speed varies inversely as the square of the diameter. The first point on the theoretic curve was taken at gage 1.3 and speed 4.3, and the curve plotted accordingly. The results of the second test were then plotted in the same scale, the points falling close to the curve as shown. The second test was taken as being the most representative. Points plotted from the third and fourth tests fell close to a similar curve. In order to compare cutting speeds of different gages, every condition, including depth of hole, etc., must be uniform, and it would not be fair to compare the speed of

a new bit with the speed of the same bit when run to a smaller gage. From this curve, it is evident that for small gages the cutting speed varies inversely as the square of the diameter of the bit. For large gages the results do not seem to check so well, and from results obtained in former work by H. Vogel, the relation of the $3/2$ power was obtained, which may apply to larger gages.

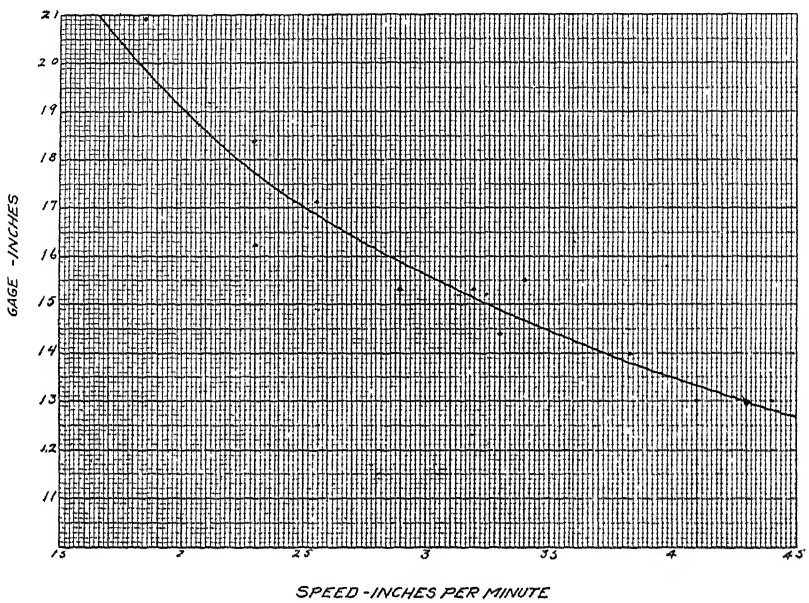


Fig. 3.—Theoretic curve assuming speed varies inversely as square of diameter, and actual results from different gages in down holes.

## Results from Up Holes

Three different pressures were used in drilling the up holes, 95 lb., 85 lb. and 70 lb. Most of the tests were made at the 95-lb. pressure. Nearly 200 different tests were made in all, but the tabulated results show only those that were most consistent.

All holes were first collared to about $1/2$ in. depth; the bit to be tested was then run at a low pressure for about 6 sec. The bit was then taken out, the gage and the depth of hole measured, and the test was ready to start. One-minute runs by stop watch were made. All bits were run for four 1-min. tests, and in a few cases six or eight 1-min. runs were made. No attempt was made to run each bit to its full extent, although in most cases at the end of four runs the cutting speed had materially diminished. The bits, however, probaby would have drilled several inches farther, and in the case of the Carr bit, considerably farther. The life of bits seems to be the same at all pressures. That is, they will drill a certain distance and no more, no matter what the pressure is.

## Results at 95 Lb. Pressure

Table 2 shows the actual results as obtained on the four different bits at 95 lb. pressure.

TABLE 2.—*Results at 95 Lb. Pressure*

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min, Inches | Average Diam | Total Dist Drilled | Average Speed Per Min | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| | | 4-POINT STARTER | | | | |
| 1 984 | 11.55 | | | | | |
| 1 920 | 14 00 | 2 45 | | | | |
| 1 908 | 16 10 | 2 10 | | | | |
| 1 908 | 18 15 | 2 05 | | | | |
| 1 908 | 20 10 | 1 95 | 1 9230 | 8 55 | 2 1370 | 0 0089 |
| 1 921 | 13 40 | | | | | |
| 1 875 | 16 00 | 2 60 | | | | |
| 1.875 | 18 30 | 2 30 | | | | |
| 1 844 | 20 40 | 2 10 | | | | |
| 1 830 | 22 20 | 1 80 | 1 8700 | 8 80 | 2.2000 | 0.0102 |
| 1 938 | 11 60 | | | | | |
| 1 858 | 14 05 | 2 45 | | | | |
| 1.844 | 16 30 | 2 35 | | | | |
| 1 812 | 18 60 | 2 30 | | | | |
| 1 812 | 20.85 | 2 25 | 1.8480 | 9.35 | 2 3370 | 0.0124 |
| 1 938 | 12.40 | | | | | |
| 1.908 | 14 90 | 2 50 | | | | |
| 1.875 | 17 20 | 2.30 | | | | |
| 1.844 | 19 25 | 2 05 | | | | |
| 1 830 | 21.10 | 1.85 | 1.8790 | 8.70 | 2 1750 | 0 0124 |
| | | 4-POINT SECOND | | | | |
| 1 750 | 18 60 | | | | | |
| 1.720 | 22.45 | 3.85 | | | | |
| 1 625 | 25 60 | 3 15 | | | | |
| 1 610 | 28 15 | 2 55 | | | | |
| 1.594 | 30.15 | 2 10 | 1 6702 | 11 65 | 2.9120 | 0 0133 |
| 1.688 | 18 80 | | | | | |
| 1.625 | 22.25 | 3 45 | | | | |
| 1.625 | 25 25 | 3.00 | | | | |
| 1.610 | 27.65 | 2.40 | | | | |
| 1.594 | 29.60 | 1.95 | 1 6290 | 10 80 | 2.7000 | 0.0087 |
| 1.750 | 19.55 | | | | | |
| 1.720 | 23.30 | 3 75 | | | | |
| 1.658 | 26 65 | 3.35 | | | | |
| 1.625 | 29.35 | 2.70 | | | | |
| 1.625 | 31.60 | 2.25 | 1.6560 | 12 05 | 2.9820 | 0.0104 |

TABLE 2.—*Results at 95 Lb. Pressure.*—(*Continued*)

4-POINT THIRD

| Gage at Start, Inches | Meas of Hole, Inches | Dist. Drilled Per Min., Inches | Average Diam | Total Dist Drilled | Average Speed Per Min. | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1.437 | 31.35 | | | | | |
| 1.375 | 35 95 | 4 60 | | | | |
| 1 375 | 40 00 | 4 05 | | | | |
| 1.359 | 43.40 | 3 40 | | | | |
| 1 328 | 46 60 | 3 20 | 1.3732 | 15.25 | 3 8125 | 0.0071 |
| 1.453 | 32 30 | | | | | |
| 1.422 | 37 20 | 4 90 | | | | |
| 1 390 | 41 35 | 4 15 | | | | |
| 1 359 | 44 95 | 3 30 | | | | |
| 1 359 | 47.90 | 2.95 | 1 3999 | 15.30 | 3 8250 | 0 0061 |
| 1.437 | 32 10 | | | | | |
| 1.375 | 37 05 | 4 95 | | | | |
| 1 344 | 41.30 | 4.25 | | | | |
| 1.312 | 44.80 | 3 50 | | | | |
| 1.312 | 47 90 | 3 10 | 1 3576 | 15.80 | 3.9500 | 0.0078 |
| 1.469 | 30.05 | | | | | |
| 1.437 | 34 80 | 4.75 | | | | |
| 1.406 | 38 90 | 4 10 | | | | |
| 1.390 | 42.35 | 3 45 | | | | |
| 1.375 | 45 35 | 3.00 | 1.4117 | 15 30 | 3.8250 | 0.0061 |

6-POINT STARTER

| Gage at Start, Inches | Meas of Hole, Inches | Dist. Drilled Per Min., Inches | Average Diam | Total Dist Drilled | Average Speed Per Min. | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1.938 | 11.60 | | | | | |
| 1.894 | 13.85 | 2.25 | | | | |
| 1.858 | 15 60 | 1.95 | | | | |
| 1.844 | 17.55 | 1.75 | | | | |
| 1.844 | 19.05 | 1.50 | 1.8660 | 7.45 | 1.8620 | 0.0126 |
| 1.970 | 12.20 | | | | | |
| 1.920 | 14.35 | 2.15 | | | | |
| 1.875 | 16.20 | 1.85 | | | | |
| 1.830 | 17.95 | 1.75 | | | | |
| 1.830 | 19.55 | 1.60 | 1.8380 | 7.35 | 1.8830 | 0.0190 |
| 1.938 | 12.80 | | | | | |
| 1.894 | 15.05 | 2.25 | | | | |
| 1.858 | 16.80 | 1.75 | | | | |
| 1 844 | 18.40 | 1.60 | | | | |
| 1.830 | 19.85 | 1.45 | 1.8740 | 7.05 | 1.7620 | 0.0153 |

TABLE 2.—*Results at 95 Lb. Pressure.*—(*Continued*)

6-POINT STARTER.

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min., Inches | Average Diam. | Total Dist. Drilled | Average Speed Per Min. | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 2.002 | 12.15 | | | | | |
| 1.938 | 14.30 | 2.15 | | | | |
| 1.908 | 16.30 | 2.00 | | | | |
| 1.875 | 18 15 | 1.85 | | | | |
| 1 858 | 19 75 | 1.60 | 1.8970 | 7.60 | 1.9000 | 0.0289 |

6-POINT SECOND

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min., Inches | Average Diam. | Total Dist. Drilled | Average Speed Per Min. | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1.720 | 18 60 | | | | | |
| 1.688 | 21.55 | 2 95 | | | | |
| 1.641 | 24 25 | 2 70 | | | | |
| 1.625 | 26 80 | 2.55 | | | | |
| 1.625 | 29.00 | 2 25 | 1.6680 | 10.45 | 2.6120 | 0.0091 |
| 1.720 | 18.55 | | | | | |
| 1 672 | 21.25 | 2 70 | | | | |
| 1.641 | 23 80 | 2.55 | | | | |
| 1.625 | 26.15 | 2 35 | | | | |
| 1.625 | 28 30 | 2 15 | 1.6550 | 9.75 | 2.4370 | 0.0097 |
| 1.750 | 19 50 | | | | | |
| 1.704 | 22 40 | 2 90 | | | | |
| 1 658 | 25 15 | 2 75 | | | | |
| 1.625 | 27.75 | 2 60 | | | | |
| 1.610 | 30.10 | 2 35 | 1.6800 | 10.60 | 2.6500 | 0.0132 |
| 1 735 | 20 00 | | | | | |
| 1 672 | 22 70 | 2.70 | | | | |
| 1.641 | 25.25 | 2.55 | | | | |
| 1.625 | 27.55 | 2 30 | | | | |
| 1.625 | 29.50 | 1.95 | 1.6440 | 9.50 | 2.3750 | 0.0116 |

6-POINT THIRD

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min., Inches | Average Diam. | Total Dist. Drilled | Average Speed Per Min. | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1.485 | 39.55 | | | | | |
| 1.453 | 42.85 | 3.30 | | | | |
| 1.437 | 46.00 | 3.15 | | | | |
| 1.422 | 48.85 | 2.85 | | | | |
| 1.422 | 51.15 | 2.30 | 1.4444 | 11.60 | 2.9000 | 0.0054 |
| 1.437 | 28.55 | | | | | |
| 1.375 | 31.75 | 3.20 | | | | |
| 1.375 | 34.80 | 3.05 | | | | |
| 1.375 | 37.75 | 2.85 | | | | |
| 1.344 | 40.40 | 2.65 | 1.3713 | 11.75 | 2.9375 | 0.0079 |

TABLE 2.—*Results at 95 Lb. Pressure.*—(*Continued*)

6-POINT THIRD

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min, Inches | Average Diam. | Total Dist Drilled | Average Speed Per Min | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1.469 | 33 35 | | | | | |
| 1 437 | 36 45 | 3 10 | | | | |
| 1 422 | 39 30 | 2 85 | | | | |
| 1.422 | 41 30 | 2 00 | | | | |
| 1 406 | 43 10 | 1.80 | 1 4327 | 9 75 | 2 4375 | 0 0064 |
| | | | | | | |
| 1 500 | 32 10 | | | | | |
| 1 469 | 34 95 | 2 85 | | | | |
| 1 453 | 37 75 | 2 80 | | | | |
| 1 437 | 40 35 | 2.60 | | | | |
| 1.422 | 42 40 | 2 05 | 1 4571 | 10 30 | 2.5783 | 0 0075 |

CARR BIT

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min, Inches | Average Diam. | Total Dist Drilled | Average Speed Per Min | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1.547 | 20 80 | | | | | |
| 1 547 | 25 00 | 4.20 | | | | |
| 1 547 | 28.30 | 3 30 | | | | |
| 1.532 | 31.20 | 2 90 | | | | |
| 1 516 | 33 75 | 2.55 | | | | |
| 1 510 | 36 30 | 2 55 | | | | |
| 1 500 | 38.80 | 2.50 | | | | |
| 1 490 | 41 20 | 2 40 | 1 5408 | 12 95 | 3.2400 | 0.0028 |
| | | | | | | |
| 1 547 | 28.45 | | | | | |
| 1 547 | 32 30 | 3 85 | | | | |
| 1.532 | 34 80 | 2 50 | | | | |
| 1.510 | 37 45 | 2.55 | | | | |
| 1 510 | 40 45 | 3.00 | | | | |
| 1 510 | 43 00 | 2 55 | | | | |
| 1.485 | 45 50 | 2.50 | | | | |
| 1.485 | 48.00 | 2.50 | 1.5300 | 11.90 | 2.9770 | 0.0032 |
| | | | | | | |
| 1.621 | 12.55 | | | | | |
| 1.609 | 16.15 | 3.60 | | | | |
| 1.594 | 20 00 | 3.85 | | | | |
| 1.594 | 22.95 | 2.95 | | | | |
| 1.578 | 25.70 | 2 75 | 1.6010 | 13.15 | 3.2900 | 0.0042 |
| | | | | | | |
| 1.623 | 13.10 | | | | | |
| 1.609 | 16.50 | 3.40 | | | | |
| 1.609 | 20.20 | 3.70 | | | | |
| 1.609 | 23 70 | 3 50 | | | | |
| 1.594 | 26.70 | 3.00 | 1.6090 | 13.60 | 3.4000 | 0.0021 |

TABLE 2.—*Results at 95 Lb. Pressure.*—(*Continued*)

Z BIT

| Gage at Start, Inches | Meas of Hole, Inches | Dist. Drilled Per Min, Inches | Average Diam | Total Dist Drilled | Average Speed Per Min | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| 1 688 | 14 10 | | | | | |
| 1 641 | 17 25 | 3.15 | | | | |
| 1.610 | 19 65 | 2 40 | | | | |
| 1 594 | 21 80 | 2 15 | | | | |
| 1.594 | 23 70 | 1 90 | 1 6160 | 9.60 | 2 3920 | 0 0098 |
| 1.672 | 13 50 | | | | | |
| 1 625 | 16 90 | 3 40 | | | | |
| 1 594 | 19.20 | 2 30 | | | | |
| 1 562 | 21.40 | 2 20 | | | | |
| 1.562 | 23.35 | 1 95 | 1 6010 | 9 85 | 2 4650 | 0 0111 |
| 1.501 | 13.60 | | | | | |
| 1.485 | 17 60 | 4 00 | | | | |
| 1 485 | 20 95 | 3 35 | | | | |
| 1 469 | 23 95 | 3 00 | | | | |
| 1.469 | 26 50 | 2 55 | 1 4800 | 12.90 | 3.2280 | 0 0025 |
| 1 469 | 14.10 | | | | | |
| 1.469 | 17.80 | 3.70 | | | | |
| 1.375 | 21.05 | 3 15 | | | | |
| 1 375 | 24.30 | 3.25 | | | | |
| 1.375 | 26.95 | 2.65 | 1.4130 | 12.75 | 3 1960 | 0 0073 |

Table 3 represents the summary of averages shown in Table 2, and also the same averages reduced, for the purpose of comparison, to uniform diameters by assuming that the cutting speed varies inversely as the square of the diameter.

TABLE 3.—*Summary of Results at 95 Lb. Pressure*

Reduced to uniform gages, assuming speed varies as $\frac{1}{D^2}$

| | Kind of Bit | Average Diam., Inches | Average Dist. Drilled | Average Speed Per Min | Average Initial Speed | Average Final Speed | Average Loss in Gage Per Inch |
|---|---|---|---|---|---|---|---|
| A | 4-point starter | 1.880 | 8.85 | 2 212 | 2.500 | 1 962 | 0.0110 |
| B | 4-point second | 1 652 | 11.50 | 2.865 | 3 683 | 2.100 | 0 0108 |
| C | 4-point third | 1.385 | 15.41 | 3 853 | 4 800 | 3 062 | 0.0068 |
| | Reduced to uniform diameter | | | | | | |
| | From A | 1 750 | ..... | 2 554 | | | |
| | From B | 1.750 | ...... | 2 552 | | | |
| | From C | 1.750 | ...... | 2.415 | | | |
| | | 1.750 | ....... | 2 507 | Average of 4-point | | |
| A | 6-point starter | 1.869 | 7.36 | 1.852 | 2 200 | 1 537 | 0.0189 |
| B | 6-point second | 1.662 | 10.07 | 2 519 | 2 812 | 2 175 | 0.0109 |
| C | 6-point third | 1.426 | 10.85 | 2.713 | 3.112 | 2.200 | 0.0068 |
| | Reduced to uniform diameter | | | | | | |
| | From A | 1.750 | ...... | 2.112 | | | |
| | From B | 1 750 | ...... | 2 271 | | | |
| | From C | 1.750 | ....... | 1.802 | | | |
| | | 1.750 | ...... | 2.062 | Average of 6-point | | |
| A | Carr bit second | 1.535 | 12.42 | 3.109 | 4 025 | 2.775 | 0.0028 |
| B | Carr bit third | 1 605 | 13 37 | 3.345 | 3.500 | 2.875 | 0.0032 |
| | Reduced to uniform diameter | | | | | | |
| | From A | 1.750 | ...... | 2.394 | | | |
| | From B | 1.750 | ..... | 2.488 | | | |
| | | 1.750 | ....... | 2.441 | Average Carr bit | | |
| A | Z bit second | 1.609 | 9.72 | 2.429 | 3.275 | 1.925 | 0.0104 |
| B | Z bit third | 1.447 | 12.87 | 3.212 | 3.850 | 2.600 | 0.0049 |
| | Reduced to uniform diameter | | | | | | |
| | From A | 1.750 | ....... | 2.053 | | | |
| | From B | 1.750 | ....... | 2.196 | | | |
| | | 1.750 | ....... | 2.130 | Average Z bit | | |

It will be seen from these results that the comparative cutting speeds of the different bits at this pressure are as follows:

(1) 4-point (gage 1.75) .................... 2.51 in. per minute
(2) 6-point (gage 1.75) .................... 2.06 in. per minute
(2) Carr (gage 1.75) ....................... 2.44 in. per minute
(3) Z (gage 1.75) .......................... 2.13 in. per minute

At this pressure the 4-point bit cuts slightly faster than the Carr. It was very difficult to make the Z bit hold up at all on the high pressure, as the corners would invariably chip off.

## Results at 85 Lb. and 70 Lb. Pressure

Table 4 shows the results obtained using 85 lb. pressure and Table 5 the results at 70 lb.

Table 4.—*Results at 85 Lb. Pressure*

| Gage at Start, Inches | Meas. of Hole, Inches | Dist. Drilled Per Min., Inches | Average Diam. | Total Dist. Drilled | Average Speed Per Min. | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| **6-Point Bit** | | | | | | |
| 1.485 | 3.10 | | | | | |
| 1 453 | 6.10 | 3.00 | | | | |
| 1 437 | 8 45 | 2.35 | | | | |
| 1.422 | 10.70 | 2.25 | | | | |
| 1.422 | 12 55 | 1.85 | 1.4444 | 9.45 | 2.3670 | 0.0066 |
| **4-Point Bit** | | | | | | |
| 1 437 | 14 75 | | | | | |
| 1 422 | 18.60 | 3.85 | | | | |
| 1.390 | 21.65 | 3.05 | | | | |
| 1 375 | 24.00 | 2.35 | | | | |
| 1 359 | 26.00 | 2.00 | 1.4020 | 11.25 | 2.8240 | 0.0069 |
| **Z Bit** | | | | | | |
| 1.485 | 13.70 | | | | | |
| 1.437 | 16 90 | 3 20 | | | | |
| 1.437 | 19 90 | 2.00 | | | | |
| 1.437 | 22.80 | 2.90 | | | | |
| 1 422 | 25 35 | 2 55 | 1.4420 | 11.65 | 2.9140 | 0.0054 |
| **4-Point 4° Taper** | | | | | | |
| 1 422 | 39.00 | | | | | |
| 1.406 | 42 05 | 3 35 | | | | |
| 1.390 | 25.25 | 2.30 | | | | |
| 1.375 | 47.90 | 2 65 | | | | |
| 1.375 | 50.10 | 2.20 | 1.3950 | 11.40 | 2.8546 | 0.0041 |
| **Z Bit Filed** | | | | | | |
| 1.390 | 14.15 | | | | | |
| 1.359 | 18.20 | 4.05 | | | | |
| 1.359 | 21.95 | 3.75 | | | | |
| 1 344 | 24.45 | 3.50 | | | | |
| 1.344 | 26.90 | 2.40 | 1.3580 | 13.70 | 3.3458 | 0.0033 |
| **Carr Bit** | | | | | | |
| 1.453 | 14.15 | | | | | |
| 1.437 | 17.65 | 3.50 | | | | |
| 1.422 | 21.05 | 3.40 | | | | |
| 1.422 | 23.95 | 2.90 | | | | |
| 1.422 | 26.25 | 2.30 | 1.4310 | 12.10 | 3.0275 | 0.0026 |

TABLE 5.—*Results at 70 Lb. Pressure*

| Gage at Start, Inches | Meas of Hole, Inches | Dist Drilled Per Min, Inches | Average Diam | Total Dist Drilled | Average Speed Per Min | Loss in Gage Per Inch |
|---|---|---|---|---|---|---|
| **4-Point Bit** | | | | | | |
| 1.62 | 13.00 | | | | | |
| 1.61 | 15.45 | 2.15 | | | | |
| 1.59 | 17.15 | 1.70 | | | | |
| 1.56 | 18.65 | 1.50 | | | | |
| 1.56 | 20.05 | 1.40 | 1.5930 | 6.75 | 1.6910 | 0.0093 |
| **6-Point Bit** | | | | | | |
| 1.50 | 13.60 | | | | | |
| 1.48 | 15.15 | 1.55 | | | | |
| 1.47 | 16.40 | 1.25 | | | | |
| 1.45 | 17.55 | 1.15 | | | | |
| 1.44 | 18.60 | 1.05 | 1.4710 | 5.00 | 1.2520 | 0.0126 |
| **Carr Bit** | | | | | | |
| 1.44 | 13.60 | | | | | |
| 1.44 | 16.40 | 2.80 | | | | |
| 1.42 | 18.60 | 2.20 | | | | |
| 1.42 | 20.60 | 2.00 | | | | |
| 1.42 | 22.60 | 2.00 | 1.4320 | 9.00 | 2.2570 | 0.0016 |
| **Z Bit** | | | | | | |
| 1.477 | 17.20 | | | | | |
| 1.477 | 20.15 | 2.95 | | | | |
| 1.422 | 22.70 | 2.20 | | | | |
| 1.422 | 25.10 | 2.40 | | | | |
| 1.406 | 26.65 | 1.55 | 1.4270 | 9.10 | 2.2780 | 0.0034 |

Table 6 shows the final average reduced to three different gages by the suqare of diameter.

TABLE 6.—*Comparative Cutting Speeds of Different Bits*

Reduced to uniform diameters, assuming speed varies $\frac{1}{D^2}$

| Kind of Bit | Reduced Gage, Inches | Average Speed Per Min., 95 Lb. P. | Average Speed Per Min., 85 Lb. P. | Average Speed Per Min., 70 Lb. P. |
|---|---|---|---|---|
| 4-point | 2 00 | 1 92 | 1 39 | 0 85 |
| 6-point | 2 00 | 1 59 | 1 23 | 0 68 |
| Carr | 2 00 | 1 87 | 1 56 | 1 16 |
| Z | 2 00 | 1 63 | 1 51 | 1 16 |
| Z flat angle | 2 00 | ... | 1 54 | |
| 4-point 5° taper | 2 00 | . | 1 39 | |
| 4-point | 1 75 | 2 51 | 1 81 | 1 11 |
| 6-point | 1 75 | 2 06 | 1 61 | 0 88 |
| Carr | 1 75 | 2 44 | 2 03 | 1 51 |
| Z | 1 75 | 2 13 | 1 97 | 1 51 |
| Z flat angle | 1.75 | .. | 2 01 | |
| 4-point 5° taper | 1 75 | | 1 81 | |
| 4-point | 1 50 | 3 41 | 2 47 | 1 51 |
| 6-point | 1 50 | 2 82 | 2 19 | 1 20 |
| Carr | 1.50 | 3 32 | 2 77 | 2 06 |
| Z | 1 50 | 2 90 | 2 69 | 2 06 |
| Z flat angle | 1.50 | ... | 2.74 | |
| 4-point 5° taper | 1 50 | ... | 2 47 | |

The curves in Figs. 4 and 5 show graphically the comparative cutting speeds of the different bits at different pressures. These were plotted by using the averages from bits of 1½-in. gage.

Figs. 6, 7 and 8 show the variation in cutting speed with gage for the different bits and at the three pressures.

## Theory of Drilling Rock

According to B. F. Tillson,[1] "when rock is excavated by a drill bit three applications of forces seem to be involved—by abrasion, by crushing, and by severing or chipping. Although all of these must take place to a certain degree, the greatest amount of useful work is performed when the percentage of force applied to chip reaches a maximum." According to this theory, the screen analysis of cuttings ought to show the most efficient bit. Samples were taken from the up holes when drilling at 95 lb. pressure, from which the screen analyses shown in Table 7 were made. Screening was done in a Tyler Ro-Tap apparatus. The samples were taken near the first part of a run with each bit. Some of the dust was lost in taking the sample, but as much as possible of the entire sample was saved.

[1] Testing and Application of Hammer Drills. *Trans.* (1915), **51**, 251.

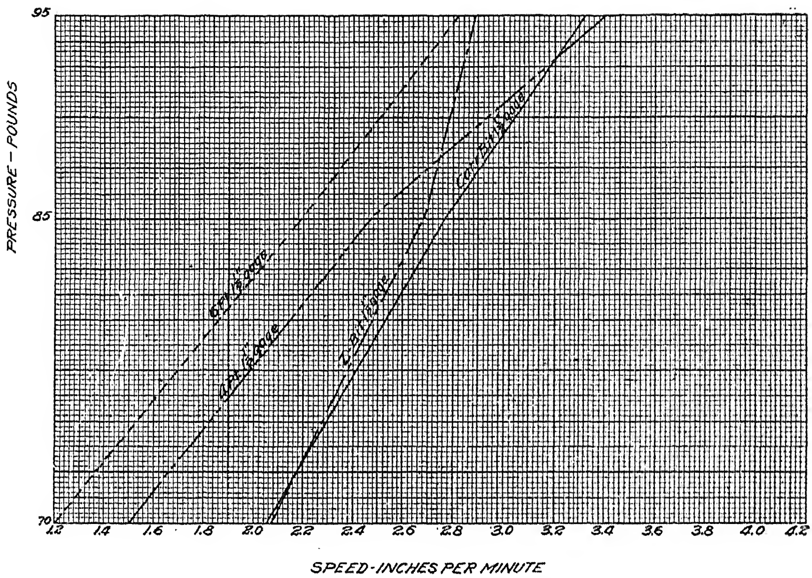


FIG. 4.—RELATIVE CUTTING SPEEDS OF 6-PT., 4-PT., CARR AND Z BITS AT DIFFERENT PRESSURES.

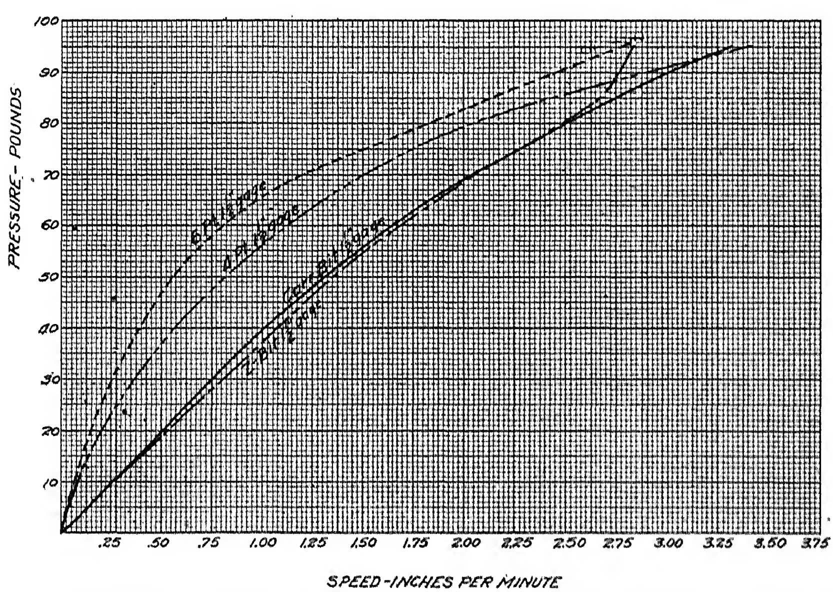


FIG. 5.—RELATIVE CUTTING SPEEDS OF 6-PT., 4-PT., CARR AND Z BITS AT PRESSURES FROM 0 TO 95 LB.

*Table 7.—Screen Analysis of Cuttings From Up Holes,* 95 *Lb. Pressure*

| On | Opening | 6-Point Bit | | Z Bit (Filed) | | Z Bit 1¾ In Diameter | | 4-Point 5° Taper | |
|---|---|---|---|---|---|---|---|---|---|
| | | Per Cent | Per Cent. Cum. | Per Cent | Per Cent Cum. | Per Cent | Per Cent Cum. | Per Cent | Per Cent Cum |
| +6 | 0 1310 | ..... | ..... | 0 69 | 0 69 | 0.62 | 0 62 | | |
| +8 | 0 0930 | ....... | ...... | 1.72 | 2.41 | 1.75 | 2.37 | 1.09 | 1 09 |
| +10 | 0.0650 | 0 59 | 0 59 | 2.76 | 5.17 | 3.75 | 6 12 | 0.73 | 1.82 |
| +14 | 0 0460 | 0 88 | 1.47 | 3.52 | 8 69 | 3.94 | 10.06 | 1.86 | 3 68 |
| +20 | 0.0328 | 5 00 | 6.47 | 3.31 | 12.00 | 4.06 | 14.12 | 2 52 | 6.20 |
| +28 | 0.0232 | 10.25 | 16 72 | 7.45 | 19.45 | 8.82 | 22.94 | 4.67 | 10 87 |
| +35 | 0.0164 | 13.82 | 30 54 | 10 00 | 29.45 | 10 18 | 33.12 | 6.68 | 17.55 |
| +48 | 0.0116 | 16.77 | 47.31 | 12 00 | 41.45 | 12.50 | 45.62 | 9.71 | 27.26 |
| +65 | 0 0080 | 14 72 | 62 03 | 11 59 | 53 04 | 12.25 | 57.87 | 14.75 | 42 01 |
| +100 | 0 0058 | 16 17 | 78 20 | 20.70. | 73.74 | 16.50 | 74 37 | 22 30 | 64.31 |
| −100 | ....... | 21 75 | 99 95 | 26.10 | 99 84 | 25.60 | 99.97 | 35.60 | 99.91 |

| On | Opening | 4-Point Bit | | 6-Point Bit | | Carr Bit 1st Run | | Dull Carr Bit | |
|---|---|---|---|---|---|---|---|---|---|
| | | Per Cent | Per Cent Cum | Per Cent | Per Cent Cum. | Per Cent. | Per Cent Cum | Per Cent | Per Cent. Cum. |
| +6 | 0.1310 | 3 05 | 3.05 | | | | | | |
| +8 | 0.0930 | 1.61 | 4.66 | 1.15 | 1.15 | 7.10 | 7.10 | 5.45 | 5.45 |
| +10 | 0.0650 | 3.26 | 7.92 | 1.78 | 2.93 | 2.42 | 9.52 | 2.72 | 8.17 |
| +14 | 0 0460 | 3.09 | 11.01 | 1.00 | 3.93 | 3 31 | 12.83 | 3.38 | 11.55 |
| +20 | 0 0328 | 3.78 | 14.79 | 2.87 | 6.80 | 2 66 | 15.49 | 0.82 | 12.37 |
| +28 | 0.0232 | 8.48 | 23 27 | 6.90 | 13.70 | 5 45 | 20.94 | 4.74 | 17.11 |
| +35 | 0.0164 | 10 22 | 33.49 | 9.90 | 23.60 | 7.10 | 28 04 | 6.97 | 24 08 |
| +48 | 0.0116 | 12 55 | 46.04 | 13.80 | 37.40 | 8.20 | 36.24 | 6.43 | 30.51 |
| +65 | 0.0082 | 14.25 | 60 29 | 17.05 | 54.45 | 12.55 | 48.79 | 15 05 | 45.56 |
| +100 | 0 0058 | 15.75 | 76 04 | 21.05 | 75.50 | 15.55 | 64.34 | 15.92 | 61 48 |
| −100 | ....... | 23 90 | 99.94 | 24.40 | 99.89 | 35.75 | 100.09 | 38 60 | 100 08 |

The curves in Fig. 9 show graphically the results of screen analysis. It will be noted that the 6-point bit produces the finest cuttings, the Z bit the next finest, the Carr next and the 4-point the coarsest. By reference to Fig. 6, it will be seen that this corresponds to the relative cutting speeds, the 6-point slowest, the Z next, the Carr next and 4-point fastest.

In order to study further the cutting action of the different bits, four shallow holes, about 1 in. deep, were drilled beside one another under the same conditions and using 85 lb. pressure.

A study of the bottoms of the holes revealed the fact that the 6-point hole was perfectly smooth, the 4-point nearly as smooth, the Z bit and Carr bit holes were quite rough, the Carr bit having a conical shaped projection in the center due to the large hole in the bit. It will be noted from Fig. 7 that the relative cutting speeds at 85 lb. pressure are as

follows: 6-point, 4-point, Z and Carr, which corresponded with the apparent roughness of the bottoms of the holes.

The use of the large hole in the center of the Carr bit is evidently one of its advantages and the same idea could be applied to advantage on all other bits. The advantage of the large hole is that it leaves a center core and diminishes the cutting surface.

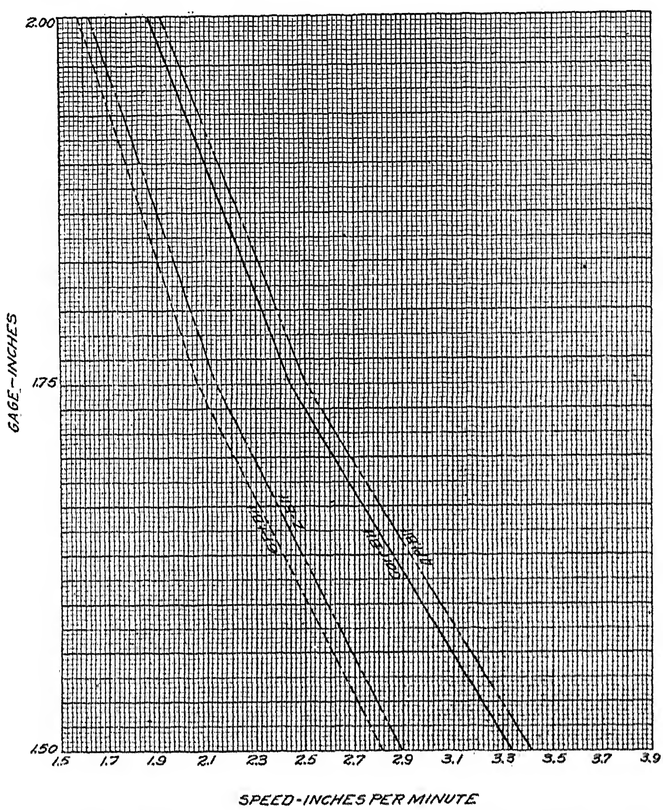


FIG. 6.—RELATIVE CUTTING SPEEDS OF 6-PT., Z, CARR AND 4-PT. BITS AT 95 LB. PRESSURE.

## CONCLUSIONS: COMPARATIVE MERITS OF EACH TYPE OF BIT

### *Cutting Speeds*

| | 95 Lb. | 85 Lb. | 70 Lb. |
|---|---|---|---|
| (1) | 4-point | Carr | Z bit |
| (2) | Carr | Z | Carr |
| (3) | Z | 4-point | 4-point |
| (4) | 6-point | 6-point | 6-point |

From the above comparison, the 6-point bit is evidently slower drilling under all conditions. The 4-point, although ranking first at 95 lb. pressure, under ordinary conditions would drill less rapidly than either the

Carr or Z. In comparison with the others, the Z bit apparently increases in cutting speed as the pressure is decreased. This would indicate that the Z bit would be quite efficient in soft rock or at lower pressures,

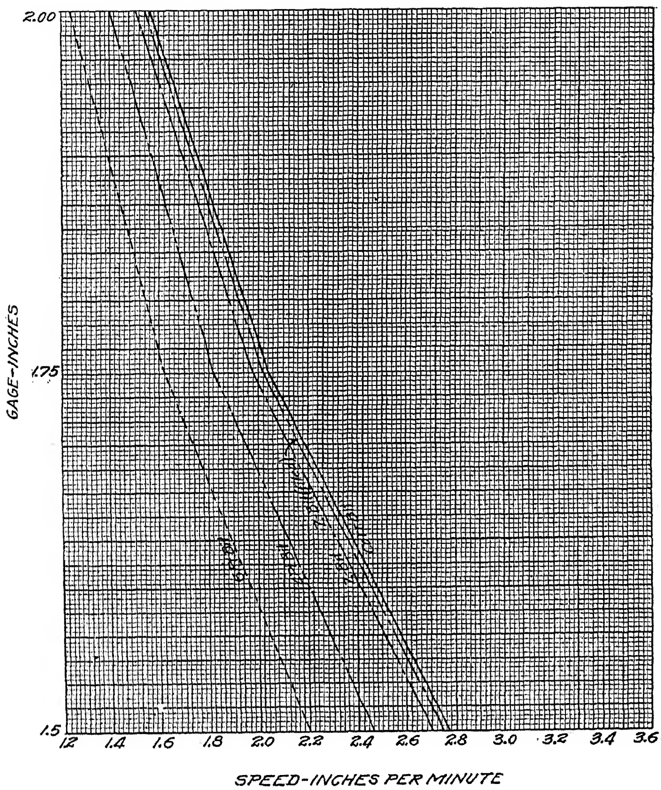


FIG. 7.—RELATIVE CUTTING SPEEDS OF 6-PT., 4-PT., Z, Z BIT (FILED) AND CARR BITS AT 85 LB. PRESSURE.

but it is evidently not adapted to extreme high pressures in hard rock. From the standpoint of cutting speed, the Carr and Z bits are the most efficient.

*Loss of Gage*

The average loss in gage per inch as shown from Tables 2 and 3, was as follows:

| | | |
|---|---|---|
| 4-point | seconds | 0.010 |
| 4-point | thirds | 0.007 |
| 6-point | seconds | 0.010 |
| 6-point | thirds | 0.007 |
| Z bit | thirds | 0.004 |
| Carr bit | seconds | 0.003 |
| Carr bit | thirds | 0.003 |
| 4-point | 5° taper thirds | 0.004 |
| Z bit (filed) | thirds | 0.003 |

This loss is almost constant at all pressures. It will be seen from these figures that the loss in gage with 4- and 6-point bits is considerably more than with Z or Carr bits, and that the 5° taper on a 4-point bit greatly diminishes the loss of gage.

From the standpoint of loss of gage, the Carr, Z bit and 4-point bit with a 5° taper, are superior to others. This factor is one that is often

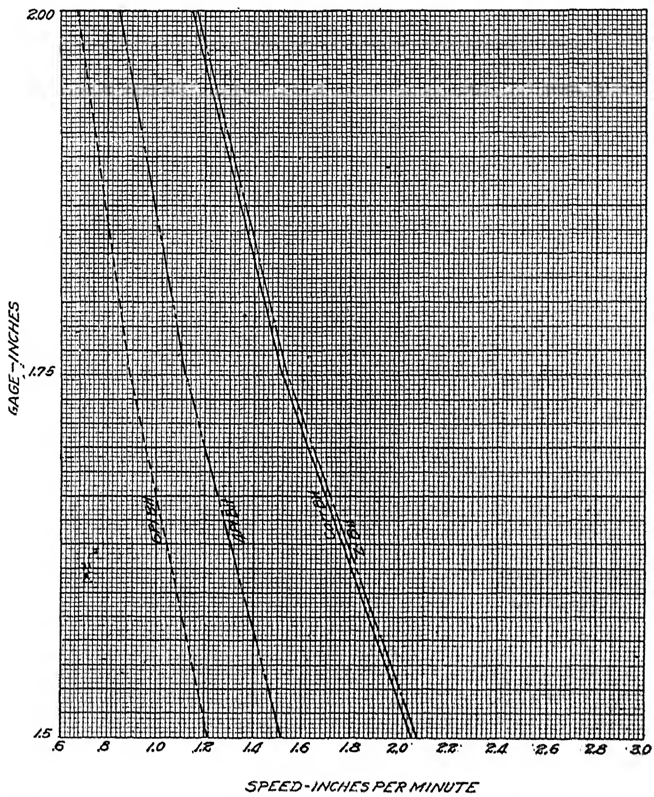


FIG. 8.—RELATIVE CUTTING SPEEDS OF 6-PT., 4-PT., CARR AND Z BITS AT 70 LB. PRESSURE.

overlooked, and the great advantage of using a bit that loses little in gage is not generally considered. As an example, the following calculation has been made:

*Drilling Time*

Drilling time for 6-ft. hole in granite, 1½-in. diameter at bottom. Length of changes, 1 ft. Difference in gages, 1/16 in. with Carr and Z bits, ⅛ in. with 4-point and 6-point bits.

CARR

| Length of Steel | 1 Ft. | 2 Ft. | 3 Ft. | 4 Ft. | 5 Ft. | 6 Ft. |
|---|---|---|---|---|---|---|
| Gage in sixteenths | $1\frac{14}{16}$ | $1\frac{13}{16}$ | $1\frac{12}{16}$ | $1\frac{11}{16}$ | $1\frac{10}{16}$ | $1\frac{9}{16}$ |
| Gage in tenths | 1.875 | 1.812 | 1.750 | 1.688 | 1.625 | 1.562 |
| Distance per minute | 1.771 | 1.897 | 2.034 | 2.186 | 2.352 | 2.554 |
| Minutes for each foot | 7.340 | 6.270 | 5.890 | 5.490 | 5.102 | 4.700 |

Time 34 min. 49 sec.

Z BIT

| | | | | | | |
|---|---|---|---|---|---|---|
| Gage | 1.875 | 1.812 | 1.750 | 1.688 | 1.625 | 1.562 |
| Distance per minute | 1.723 | 1.839 | 1.978 | 2.126 | 2.295 | 2.484 |
| Minutes for each foot | 6.960 | 6.530 | 6.070 | 5.640 | 5.230 | |

Time 35 min. 16 sec.

4-POINT

| | | | | | | |
|---|---|---|---|---|---|---|
| Gage | $2\frac{1}{4}$ | $2\frac{1}{8}$ | 2 | $1\frac{7}{8}$ | $1\frac{6}{8}$ | $1\frac{5}{8}$ |
| Distance per minute | 1.009 | 1.230 | 1.389 | 1.580 | 1.814 | 2.104 |
| Minutes per foot | 11.890 | 9.760 | 8.640 | 7.590 | 6.620 | 5.700 |

Time 50 min. 12 sec.

6-POINT

| | | | | | | |
|---|---|---|---|---|---|---|
| Gage | $2\frac{1}{4}$ | $2\frac{1}{8}$ | 2 | $1\frac{7}{8}$ | $1\frac{6}{8}$ | $1\frac{5}{8}$ |
| Distance per minute | 0.976 | 1.092 | 1.235 | 1.405 | 1.618 | 1.867 |
| Minutes per foot | 12.295 | 11.000 | 9.700 | 8.540 | 7.440 | 6.430 |

Time 55 min. 24 sec.

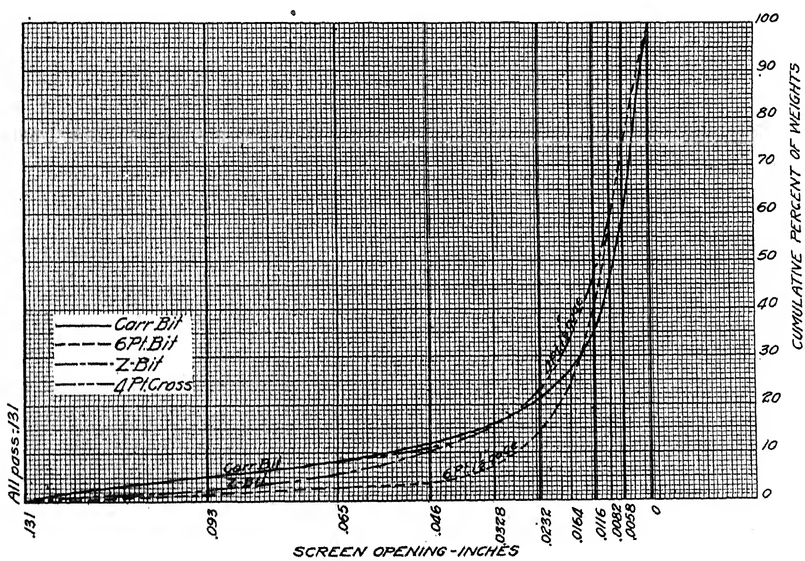


FIG. 9.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON CUTTINGS AT 95 LB. PRESSURE.

From these calculations the Carr and Z bits apparently save one-third in drilling time over the 4- and 6-point bits. This is due more to the fact that smaller gages can be used than to greater cutting speed. The calculations were based on actual cutting speeds and loss in gage shown in tests.

### *Breakage and Ease of Sharpening and Tempering*

There is little difference between the 4-point and 6-point bits in respect to ease of sharpening and tempering.

The Z bit is more easily made, but it is very difficult to temper so that it will stand up under high pressure. This is on account of the weakness of the outer cutting edges.

The Carr bit is by far the easiest of all to make and temper, and can be used in a much harder condition than others.

## Summary

1. The results in down holes indicate that the cutting speed varies inversely as the square of the diameter, at least for smaller gages.

2. Drilling speed increases almost uniformly with increase in pressure, as shown in Fig. 3. A pressure of about 85 lb. per square inch seems to be best adapted to all bits for drilling in rock of the hardness of that used in the tests.

3. Speed of drilling seems to be proportional to the coarseness of the cuttings, as shown by screen analysis and study of the bottoms of the drill holes.

4. Taking into consideration its cutting qualities, loss in gage, ease of making and tempering, the Carr bit seems to be far superior to all others, except possibly the Z when used at low pressures.

The Z bit at low pressures and probably in soft rock would equal if not surpass the Carr bit in cutting speed, but on account of the difficulty in its making and tempering, it is doubtful whether it would be as desirable under any conditions as the Carr.

For exceedingly high pressures in very hard rock, the 4-point bit made with a 5° taper on the wings seems to be superior to all others.

The 6-point bit apparently has little to recommend it in any circumstances, although it is convenient for starting holes.

# Mine Models

BY H H. STOEK,* E. M., URBANA, ILL.

(St. Louis Meeting, October, 1917)

MINE models have three distinct uses:

1. As exhibits in expositions and museums.
2. As exhibits in law suits.
3. As illustrations in teaching mining engineering.

All three uses are in a sense educational. The third, or distinctively teaching function, has not been extensively developed in America as it has been abroad, chiefly because of the cost of the models and the scarcity of model makers.

In connection with expositions, such as the Columbian, the Louisiana Purchase, and the Panama-Pacific, a number of very good mine and metallurgical models were prepared with funds furnished by State commissions, by mining organizations or mining companies. At the close of these expositions, some of these models have been deposited in National or State museums or in educational institutions, but too many of them have been lost, or have been stored away in some mining company's office or in the basement of some State building, where they have been lost sight of and have gradually fallen to pieces.

A visitor to a German mining school is impressed by two facts:

1. The mining laboratories are in general inferior to those in America.
2. The mining museums are infinitely superior to ours, particularly in models exhibited. One may reply to this comparison by saying that a museum is necessarily a work of time. From the historical standpoint this is, of course, true, but it is not necessarily true in connection with a working museum intended to illustrate current practice. For instance, the oldest mining school in Germany, that at Freiberg, has much excellent historical material, but very little on modern practice, while Berlin, one of the youngest German mining schools, has a most extensive and valuable collection of modern appliances and models. This is true in spite of the fact that one of the best-known model shops is at Freiberg, from which many of the models to be seen in America have come. Because many of the models in American mining schools have come from

* Professor of Mining Engineering, University of Illinois.

this shop, they mainly illustrate ore-mining rather than coal-mining practice. Also, the excellent models put out by the Engineering Model Works of Butte, Mont., represent almost entirely geological problems or ore-mining practice.

Because it has been impossible to obtain foreign models, and' on account of the prohibitive charges of model makers in the Middle West, who confine their attention mainly to mechanical subjects, the Mining Department of the University of Illinois is attempting to develop in its own shop a series of models to illustrate the typical methods of mining. Thus far those built have referred to coal-mining practice and have been constructed mainly of wood. This material offers certain difficulties not met with in connection with plaster models, and as some of these difficulties seem to have been overcome, the following description is offered in the hope that it may possibly assist others who are attempting to solve the educational model problem and also that others may give their experiences along similar lines.

*Baseboard.*—It was foreseen that in models built of wood contraction and cracking would be serious items, as was illustrated by a wooden model bought in Freiberg some years ago, and which, after being at Illinois for 25 years, still contracted. An effort was, therefore, made to build up a base for the model upon the principle of a well constructed drafting board. The foundation for the relief portion of the models consists of a baseboard of kiln-dried soft pine, 5 ft. square, made up of 6 by 1⅛-in. stock held together by four 1¾ by 3½-in. oak strips, evenly spaced. Considerable trouble was experienced at first by the board shrinking and cracking, and in an effort to obviate this, slots were provided in the oak strips through which the screws pass to the baseboards proper, washers being placed under the screw heads. This construction permits a movement or sliding of the board components upon the strips, and on several of the models some shrinkage has evidently taken place without cracking. On one of those first constructed, however, a few slight cracks have appeared, but these can be easily filled with putty, and repainted without injuring the model. The climate in central Illinois is very hard on any construction of this nature.

Upon the top of this large baseboard was placed a blue print of the layout of the mine workings, each pillar being numbered. Then by placing carbon paper underneath the blue print and tracing over it, a reproduction of the mine plan was left on the board, all parts being numbered as on the blue print.

The "coal" pillars and coal in place were built up of kiln-dried white pine, the thickness of the board depending upon the thickness of the seam and the scale of the model. It is usually necessary to exaggerate the vertical scale. The blue print plan was pasted on the pine board and the numbered small blocks representing the pillars and solid coal

cut out with a jig saw. The large board was first cut up into smaller sections for easy handling.

The separate pieces were then given a coat of hot cabinet glue on the bottom, placed in position on the baseboard according to number, and at least two ⅝-in. brads driven through each block and countersunk. The larger pieces required more nailing to hold them down. With a pocket knife the rooms and entries were trimmed, so as to make them as realistic as possible. After this trimming, the whole surface was

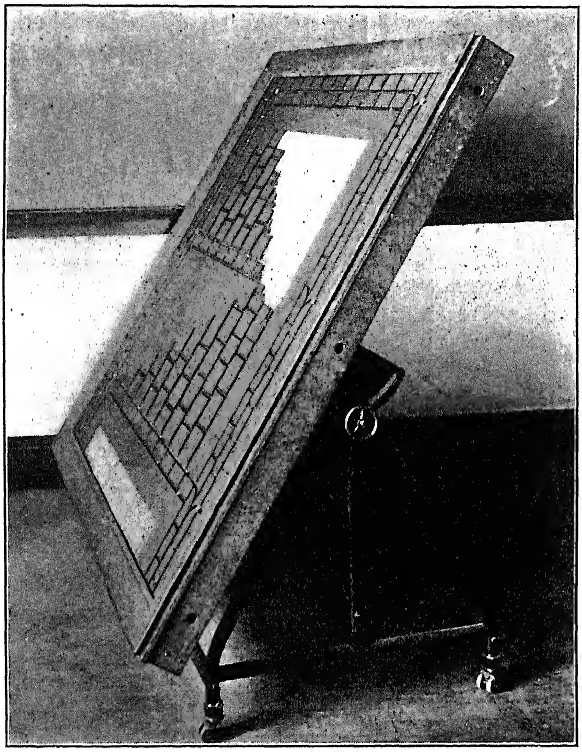

FIG. 1.—PILLAR-DRAWING MODEL AND MODEL FRAME.

sandpapered down to a uniform level. A large block covered with sandpaper was used for the first models, but later a revolving circular sandpaper-covered block propelled by an electric motor and flexible shaft reduced the labor and time by at least two-thirds.

The entire surface, including the underside of the baseboard, was then given a coat of drab paint, followed when dry by a coat of black on the upper surface. Nail holes were then puttied and smoothed.

The appearance of bituminous coal was secured by first applying a coat of black paint on top of the relief portions only (but not to the sides and bottom of the rooms and entries), then sprinkling upon the fresh

paint a layer of coal dust. Anthracite dust proved the best to give a permanent luster and uniform color. When the paint had dried sufficiently, any superfluous coal dust was dumped from the board, leaving a smooth layer firmly fastened to the surface of the relief and giving it a very satisfactory coaly appearance.

Overcasts were made of sheet iron, formed into a rectangular hollow section by soldering covers on troughs, cutting to required lengths, and filing to the exact shape desired. These overcasts were glued on wooden abutments and the whole painted drab to resemble concrete.

Worked-out sections, where roof had caved, were filled nearly to the level of the top of the coal seam with broken shale firmly glued together.

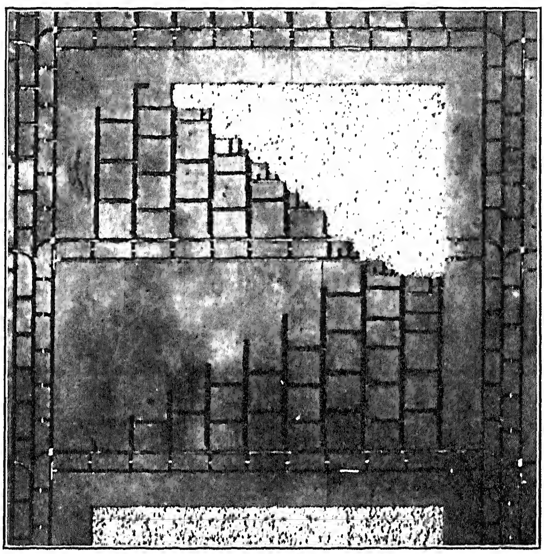

FIG. 1a.—NEARER VIEW OF PILLAR-DRAWING MODEL SHOWN IN FIG. 1.

The light portions in Figs. 1 and 1a, where the pillars have been drawn, show the contrast between the broken rock and the coal.

In the longwall model (Fig. 2) the pack walls were built up for half their vertical thickness with wood strips, which were then veneered with thin flat pieces of shale of fairly uniform size and thickness, that were glued to the wood strips. At first it was very difficult to make the glue stick permanently to the wood, for with the shrinkage of the board, sections of glue and shale would loosen and pop off. This was overcome by cutting holes in the wood strips by means of a double tracing wheel with blunted points, thus giving the glue an additional anchorage. A number of schemes for the rapid laying of these pack walls were tried, but the only effective way to produce a natural-looking wall was to lay the pieces one at a time with tweezers, suitable pieces being first selected

from the mass of crushed stone and placed in a shallow pan. This was a slow process, but results have justified the method. Gob between the pack walls was filled in by pouring large and small fragments into a layer of glue.

Mine track is represented by tacking along the entries and in the rooms copper wire screen cloth cut into strips one square wide. As screen wire cloth is not soldered, it will fly to pieces when cut in narrow strips. This was avoided by dipping wider strips into molten solder before cutting. Tracks were placed before the coal dust was applied. A neat method of fastening down tracks, which was used on the last two models, was to drive nails with solder-dipped heads flush with the bottoms of entries and about 4 in. apart. A hot soldering iron was then used to unite the track with the nail heads. Curves were bent to required radius before setting.

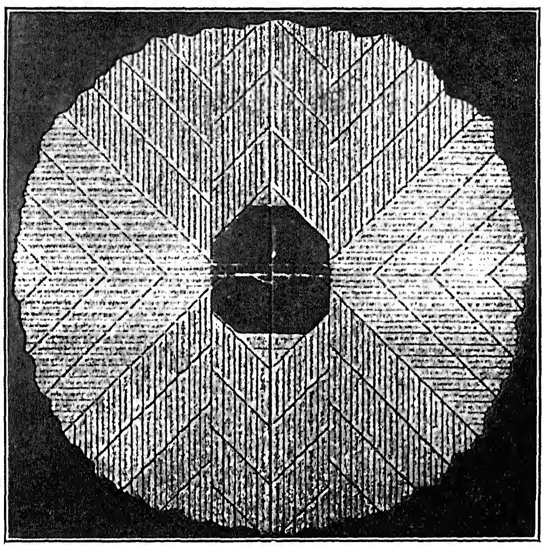

Fig. 2.—Longwall model.

Arrows placed in the entries show the direction of the air currents, white arrows representing fresh air courses and red arrows return courses. Doors and regulators are made of thin wood, painted. By attaching pins to the overcasts, doors, regulators and ventilation arrows, they can be made temporary and movable and different methods of carrying the air can be illustrated, so that the model can be used instead of a map for working out with a class problems in ventilation.

The frames are of oak picture molding, about 4½ in. wide, supplemented by oak pieces placed at right angles to the molding so as to hide the baseboard. These sides are mitered and secured by brass screws.

A title is placed at the top of each frame.

The supporting stands, Fig. 1, are built of 1-in. pipe, have double-wheeled ball-bearing castors, and are equipped with a quadrant and

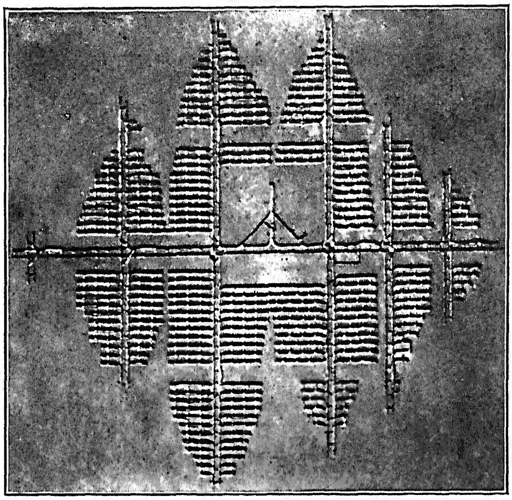

Fig. 3.—Room and pillar model.

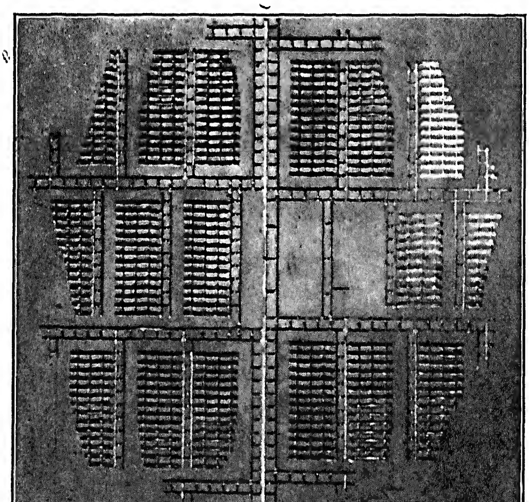

Fig. 4.—Panel room and pillar model.

hand screw, as shown, so that the model may be inclined at any desired angle—from horizontal to nearly vertical. When inclined, the width over all is such that the model can be taken through any door in the mining building.

In the development of these models great credit is due L. S. Baldwin, at present instructor in engineering drawing at the University of Illinois, who as an undergraduate student at the University did all of the detailed work on the models and developed most of the methods of construction, and also to H. J. Vanderbeek, mechanician of the Mining Department, University of Illinois, who designed and built the supporting stand for the models.

Thus far, four models have been built to illustrate the three typical methods of coal mining, namely, simple room and pillar, Fig. 3; panel room and pillar, Fig. 4; longwall, Fig. 2; and the method of drawing pillars in the panel system where narrow rooms and wide pillars are used, Fig. 1.

## Discussion

Joseph Daniels, Seattle, Wash. (communication to the Secretary*).—Mine models are of great value to the student and to the layman in visualizing underground conditions, and, in a measure, should be just as much a part of the equipment of a mining school as sets of ore specimens, catalogs, or drawings. The chief drawback, as Professor Stoek has indicated, is the time and cost of making them. The greatest good in the development of the model comes to the person who makes it, and I have often wondered whether it might not be possible to have each student make one model during his undergraduate career. At the University of Washington we have been collecting sets of mine timbers, made of 6 by 6-in. lumber, illustrating shaft, level, and stope timbering. This work has been done by freshmen.

Perhaps each educational institution could specialize in the creation of a certain type of model and exchange its product for that of another school, just as we now do with ore collections. In this way we could build up good museums at a relatively low cost.

John R. Chamberlin, New York, N. Y. (communication to the Secretary†).—The type of model described by Professor Stoek is admirably suited to exhibits of coal mining in horizontal veins, and fulfills perfectly the three purposes mentioned. As an educational illustration in colleges, especially when the students themselves build the model under the direction of their instructors, it is of the highest value; and as each condition in mining requires special methods to depict properly in a model, the student is compelled to exercise considerable inventive resource to construct a true representation, stimulating the imagination and interest in his subject.

Models built for exhibits in lawsuits are usually constructed to show prominently some particular feature of vein formation or geology, or

* Received May 17, 1917 † Received June 26, 1917.

the relation of underground workings to surface property lines, and as such their use is invaluable to the successful litigant. When the litigation is ended, it is natural that the model should be relegated to the basement and frequently destroyed, as data on underground workings are often considered of a private character by the operators; and it is rare and fortunate when such a model may be preserved for the education of young engineers.

Many mining companies have attempted, with more or less success, to keep up a progress model of underground workings. In the models described by Professor Stoek this is comparatively easy, but in ore mining or dipping veins and a complicated vein system, where the use of a model should be most instructive, the work of making additions is frequently such a delicate and "puttering" job that current developments are often allowed to fall behind and the model fails to serve its purpose. Where many levels are to be shown, it is customary to use glass sheets held in place by more or less elaborate framework, and perhaps some important between level workings shown with celluloid strips. When it comes to taking such a model apart for the purpose of making addition, the engineers and draftsmen are often inclined to lose interest and neglect it, especially as the practical part of their work is better served by the mine drawings on a working scale, which furnish them a much more intimate knowledge of underground conditions than a model. To the operating staff at the mine, a model is generally considered a clever bit of ingenuity, but of little practical value.

To the home office, however, the model is frequently of great interest, and occupies a prominent place in the Director's room. A model for exhibition purposes, of course, may be very elaborate and as costly as the Directors will permit. The writer, several years ago, made a number of solid glass models in which the underground workings were carved out and colored, the configuration of the surface carved to scale, the sides and surfaces polished and surface improvements shown in relief. This work, of course, required special machinery and was expensive.

W. R. Crane, State College, Pa.—The use of mine models for instructional purposes, both in the classroom and for more general work, as in the engineer's office, is of so much importance that I was pleased to see Professor Stock's paper which supplements a fairly voluminous literature on the subject.

I have been interested in model making for instructional purposes for nearly 20 years and have tried my hand at practically every phase of the work, and have also employed practically every kind of material that might be suitable in model construction. I recently hit upon the idea of using clay in order that methods of mining might be shown more readily and to greater advantage before my classes. Artificial modeler's clay

has proved very useful, particularly as it does not dry out and consequently does not crack through shrinkage.

By the use of more or less elaborately made models, I can confidently say that the grade of work done in the classroom has improved in a surprising manner, which I attribute largely to the ability of the

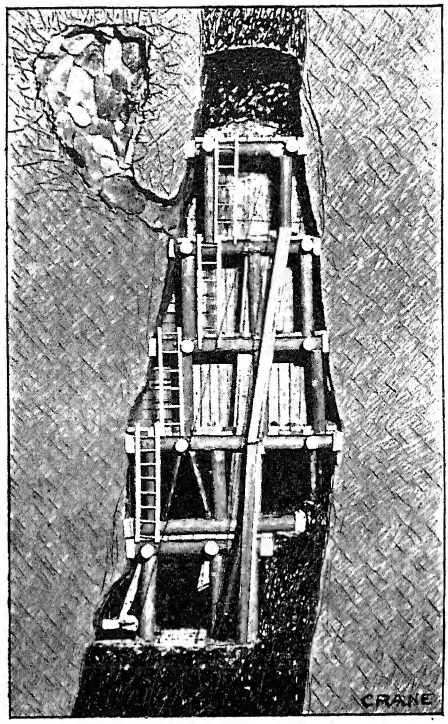


Fig. 1.—Model of a timbered stope.

student to see the mine plans and workings in their entirety and to the fact that all parts are shown in relief, thus bringing out the three dimensions.

Models have been used in the past mainly for exhibits in courts of law and in museums, but in the future they will undoubtedly be employed more extensively in the classroom and the engineer's office.

The accompanying figures show very well how models may be used to illustrate mining methods.

F. W. Sperr, Houghton, Mich.—I have been making mine models, not in great numbers, for quite a number of years past, but I felt the need of them for a long time before I began to make them. A number of questions had to be decided in the beginning: the kind of material to be used, the degree of realism to be attempted, the comprehensiveness of

the different stages of operation to be shown, the ratio of the scale to be used, and whether the representations should be static or whether attempts should be made to represent action.

My first models were made of real rock, with cement mortar and concrete; they are quite satisfactory so far as illustrating a condition, but they are too heavy, and the necessary scale with this material is so large that it was impracticable to make the models sufficiently comprehensive. Then we tried wooden models on a scale of about 1 to 100 and attempted to show all the different stages of mining from the collar of the

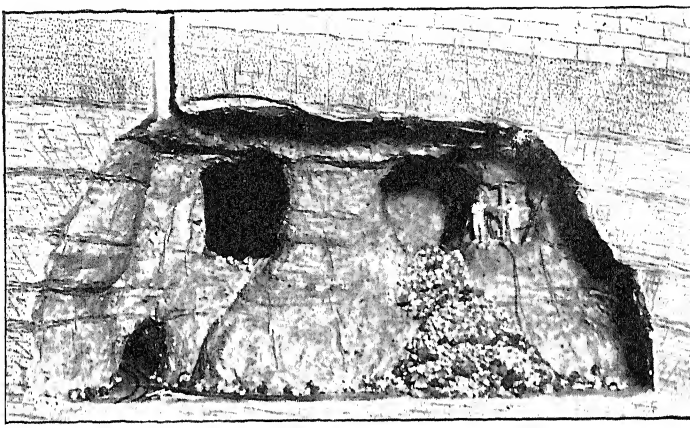

FIG. 2.—MODEL OF AN OPEN STOPE.

shaft to the completed stope, statically; in these respects, the models are satisfactory, but they are not very realistic. It is difficult to simulate rock details in wood, especially so for pattern makers and wood workers who have never been underground and cannot get the conception of what it looks like. Papier maché was tried and, I believe, would have been found very good if facilities had been available for properly baking it. The material that meets all the requirements most nearly, so far as our experience goes, is a wire base covered with magnesia pipe covering. This lends itself to the execution of small details as well as to comprehensiveness, together with a degree of realism not attainable with the use of the other materials. Lately we have been trying pasteboard, which was suggested by C. E. Pettibone, Safety Engineer for Pickands Mather & Co.; and we find it well adapted for some methods of mining. We are still looking for other materials and other ways of making mine models, and we are thankful for the valuable suggestions in Professor Stoek's paper.

We have accomplished little in the way of making models to show action in mining. In the last two or three years, we have been illustrat-

ing caving action behind plate glass and we are now aiming to get a combination of materials of a scale of strength proportional to scale of dimensions to illustrate crushing of supports, the behavior of subsidences, creeps and air blasts.

Our uses for models are purely academic, and we have found a particular way of using them most profitably. At first we thought that trips to the mines of which we had models would be almost superfluous,[1] but we soon discovered that in this we were much mistaken; we also found that having the student view the model while we explained it to him failed to increase the efficiency of our teaching as much as we had anticipated that it would do. Now, in addition to the viewing and explaining, we require the students to sketch and describe the methods of mining from the models and take their sketches along on their underground trips for convenient reference, to help them understand what is going on around them, above and below, when they are in some particular part of the mine; in this way a great deal of time is saved. One can accomplish in a few hours what might take several hours to cover without the sketches; and, in some cases, it might take years for the whole cycle of operations to take place. This is why we require comprehensiveness above everything else in our models.

Our chief difficulty, however, in making models is to find the necessary time. It is a comparatively small matter to get drawings made for the model maker, but the drawings have to be explained as the work progresses. Men have asked me to furnish them drawings from which they could have models made; but if I could make drawings of mine workings from which a model maker could make models, I would need no models.

E. B. Young, Butte, Mont.—May I say a word about a type of model built by the Engineering Model Works, of Butte, Mont., of which F. A. Linforth is manager? The method, roughly, is this: a heavy wooden base is made, strong enough to keep its shape, and on this is placed a composite map of the workings. Metal shafts and wire towers support the workings, the various levels and stopes being cut out of maple, according to the tracings. The whole model is thus open, and on it the geology can be painted in such a way as to bring out very clearly space relationships.

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[1] I believe our coal-mine model was our first. We have no coal mines to which we can conveniently make trips. Black fiber-board was used for the coal, gray fiber-board for the fire-clay, and celluloid for the slate roof, through which one can see the entries, rooms, pillars, regulators, overcasts, etc., underneath

## Mining Methods of the American Zinc Co. of Tennessee

BY H. A. COY* AND H. B. HENEGAR,† B. S., MASCOT, TENN.

(St Louis Meeting, October, 1917)

THE Mascot mines of the American Zinc Co. of Tennessee are situated in the Holston River valley, in Knox County, Tennessee, about 13 miles (20.9 km.) east of the city of Knoxville, and form a property of three operating shafts and one open pit. A second open pit is located at Jefferson City about 16 miles (25.8 km.) east of the Mascot mines.

### GEOLOGY

*Topography.*—The principal topographical features of the region within which the zinc deposits occur is a series of parallel northeast and southwest ridges which reach altitudes of 200 to 400 ft. (61 to 122 m.) above the valley. These ridges are formed of resistant formation such as sandstone, quartzites, etc., while the valleys are occupied principally by limestone, including Knox dolomite, and by shale and other formations subject to quick erosion.

The Holston River valley is wide and open with a rolling surface. Sink holes are common to the limestone portion, the general aspect of the valley being that of well-rounded hills with low intervening valleys superimposed upon which is a characteristic sink-hole topography.

*General Geology.*—According to Keith, the rock formation of the area extends from the lower Cambrian into the Silurian. The region has undergone severe deformation with heavy faulting, the underlying Cambrian having been brought to the surface by a series of low angle thrust faults, the strike of the faults and formations paralleling the general northeast and southwest direction of the topography and dipping at varying angles to the southeast. In general, the Cambrian quartzites, sandstone, etc., occupy the ridges while the Cambro-Si urian dolomites, limestone and shales occur principally in the valleys. Of the latter, the Knox dolomite is of particular interest to this paper in that it is the ore carrier in the Mascot area. The ore is associated with more or less irregular breccia zones, the degree of brecciation varying from a fine crushing to a breaking that involves blocks of large size. The breccia zones

* Supt. of Mines, American Zinc Co. of Tennessee.
† Ass't. Supt. of Mines, American Zinc Co. of Tennessee.

grade laterally and vertically into unfractured or slightly fractured ground.

The pay ore so far mined has a maximum thickness of 110 ft. (33.5 m.) and pinches down, or runs into low grade as unfractured portions of the formation are approached.

*Ore Occurrence.*—The ore occurs both as a sulphide and carbonate, the sulphide alone being mined by the American Zinc Co. The carbonate is found only in small pockets at shallow depths.

## Mining Methods

Two of the present operating shafts of the American Zinc Co. of Tennessee were sunk approximately 100 ft. (30.5 m.) south of the orebody and are on an east and west line 2800 ft. apart.

No. 1 shaft (which was the first shaft sunk by the American Zinc Co.) is located on what is known as Holston Hill. This is a three-compartment shaft consisting of two skipways 5 ft. 6 in. (1.68 m.) in the clear and one 3 ft. 6 in. (1.07 m.) cageway which has in connection two 24-in. (0.61 m.) pipeways. This shaft is square setted to solid rock ("or bottom of cribbing") with 6 by 6-in. (152.4 by 152.4-mm.) oak timbers. From this point down 8 by 8-in. (203.2 by 203.2-mm.) timbers were used, each set being blocked to the walls, and bearing sets being inserted about every 50 ft. (15 m.); 4 by 6-in. (101.6 by 152.4-mm.) dressed-pine guides are used on the skipways and 4 by 4-in. on the cageways. The guides are joined together by tongue and groove and fastened to dividers with 8-in. lag screws, the heads of which are countersunk in guides. This shaft was sunk to a total depth of 380 ft., the main station being cut at 280 ft. down. From this station a main level crosscut was driven due north, through the orebody to a point under an incline leading from an old shaft which had previously been sunk by former operators. A connection was made between main level and incline by a raise, this being done for ventilation and to give a separate manway to surface, which is one of the requirements of the mining laws of Tennessee. Drifts were driven east and west from main level following foot wall and hanging wall. At 100-ft. intervals along these drifts crosscuts were driven due north, a number of which extended as far as 300 ft. into the foot wall. At points along each crosscut, raises are driven to orebody, the spacing of these raises depending entirely upon the system laid out in advance for mining that particular ground, as each stope is more or less of a problem of its own, and has to be mined independently, especially the stopes on the upper runs which have no connection with the main run.

Several systems have been tried for mining the ore lying above the main level. The shrinkage system was tried on the east side of this mine but did not work out because the ground broke into large slabs and

choked the pull holes. Again, it was almost impossible to get a safe roof for machine men to work under and the conditions as a whole justified discontinuance of this system entirely.

The underhand system now being worked has proved to be the most satisfactory way of working this ground. With this system, an 8-ft. (2.5-m.) heading is cut along the hanging wall and the roof made safe before the bench is removed. As these headings advance, sublevel trams are established, the ore being dumped into raises which were driven in advance as stated above. As a rule, one raise takes care of

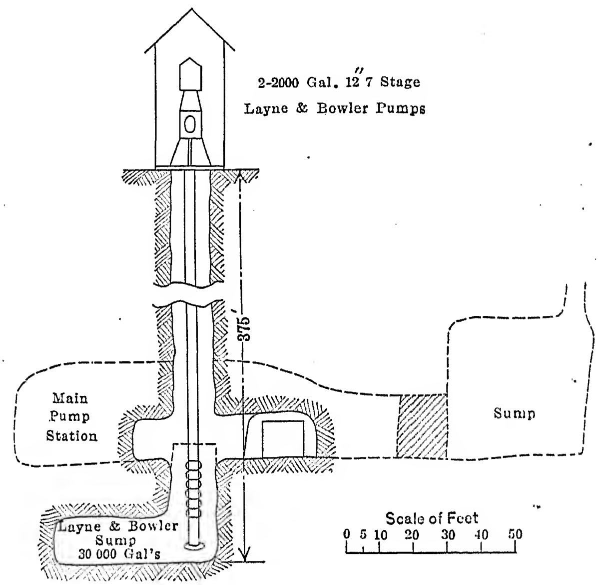



FIG. 1.—SECTION THROUGH EMERGENCY PUMP SHAFT, NO. 1 MINE.

two sublevel trams which enter at different elevations. Where there is a higher run of ore, these raises are extended and the same systems worked as on the lower runs. For entering the headings of the lower runs of ore the open stopes are used, but for the upper runs a protected manway with staggered ladderways is furnished, which also takes care of the air and water lines. Where the ore goes below the main level, inclines are put down on the foot-wall side and after cutting at lower stations, drifts are driven east and west and the same stoping system worked as on the upper ore.

No. 2 shaft is situated 2800 ft. (853 m.) due east of No. 1. This was the second shaft sunk by the American Zinc Co. and is a four-compartment shaft, consisting of two skipways and one cageway, each 5 ft. by 5 ft. 6 in. (1.53 by 1.68 m.) in the clear, and one pipe and manway 3 ft. by 5 ft. 6 in. (0.92 by 1.68 m.) in the clear. This shaft is square setted

practically the same as No. 1 with the exception that an 8 by 8-in. (203.2 by 203.2-mm.) oak timber is used for the entire distance. No. 2 shaft was sunk to a depth of 612 ft. (186.5 m.), the main station being cut on the west side at 520 ft. (158.5 m.) down. From this station a haulage drift encircles the shaft and is widened out on the north side where there are two short 8 by 12-ft. (2.5 by 3.7-m.) drifts, one of which goes due east to the head of the east incline, the other due west to the foot of the west incline. The east incline dips 20° for a distance of 350 ft. (106 m.). The west incline rises 24° for a distance of 825 ft.

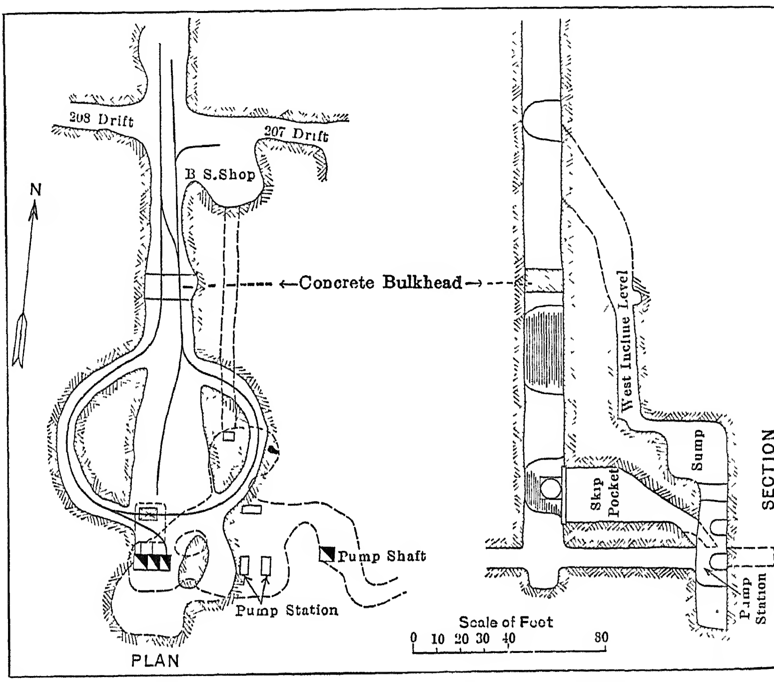



Fig. 2.—Plan of main station, No. 1 mine.

(251.5 m.). Both inclines are cut 8 ft. high and 10 ft. wide and carry a single haulage track of 40-lb. (18.2-kg.) rail, 24-in. gage, which is laid close to the south rib, giving ample room on the north side for a cleated board walkway.

The same general stoping systems are used at this mine as at No. 1, but for handling the ore above the main level the incline system is used in place of the sublevel system. At approximately 100-ft. (30-m.) intervals along both the east and west inclines, crosscuts are driven from the incline on the hanging-wall side north to the foot wall. Each crosscut is protected for tramming and stopes are opened up on the hanging-wall side of each crosscut, working in the direction of the next crosscut

which is approximately 30 ft. (9 m.) higher vertically. After the top heading is cut along the hanging wall, the bench is removed as fast as possible in order to get shoveling room on the crosscut level. The ore on the main level between the east and west inclines is removed by first cutting out the sill floor, then working the overhead system, holding the muck until the roof is made safe and a top heading is cut on hanging wall.

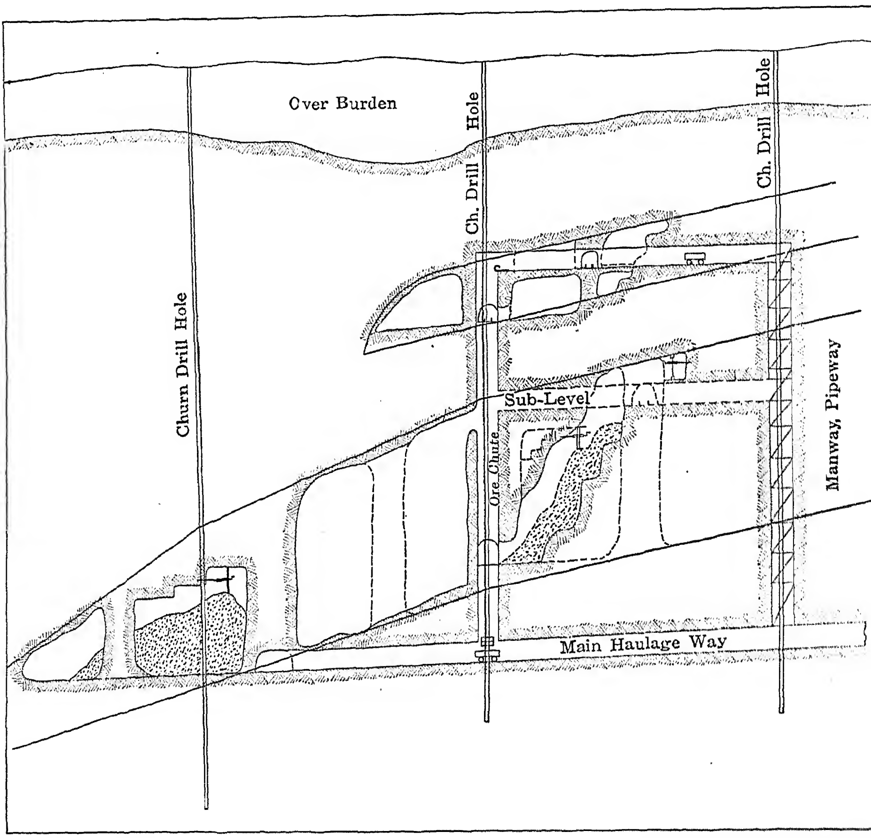



FIG. 3.—SECTION SHOWING SILL FLOOR AND SUB-LEVEL SYSTEMS OF STOPING.

No. 3 mine, which is approximately 3000 ft. (914 m.) due west of No. 1, was formerly worked by the Grasselli Chemical Co. and purchased by the American Zinc Co. in 1914. Work was at once started in making this shaft two compartments 5 ft. by 3 ft. 10 in. in the clear. Quite a lot of time was consumed in catching this shaft up, as it had caved in several years before. After this was accomplished, the shaft was square-setted and concreted to solid rock a distance of 50 ft. From this point 8 by 8-in. oak stulls were inserted in well-cut hitches. These stulls

were placed 6 ft. apart and were for the purpose of carrying in 4 by 6-in. guides.

The former operators had mined approximately 50,000 tons which had been taken out of the immediate vicinity of the shaft at a point 187

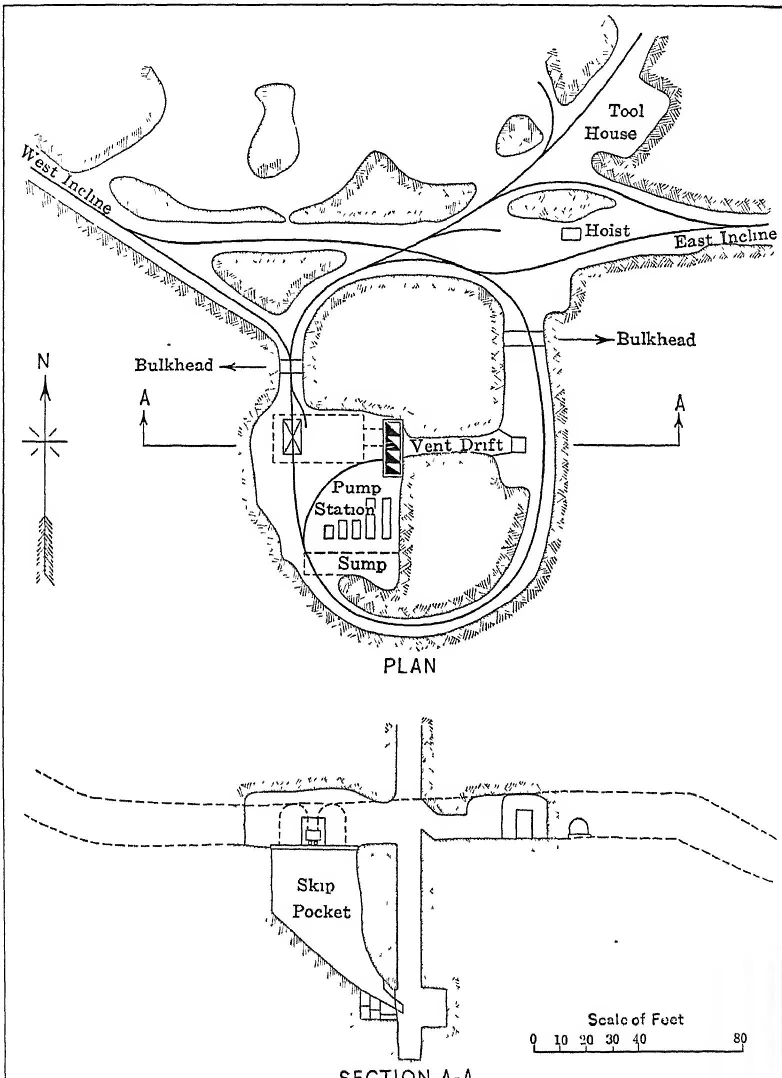



Fig. 4.—No. 2 station layout.

ft. (56.9 m.) below the collar. Previous to the purchase of this property, the American Zinc Co. drove two prospect drifts and one raise. One of these drifts was driven with the ore due east for a distance of 300 ft.

(91 m.), the other crosscutting the ore due north 100 ft. (30.5 m.); from the end of this north crosscut a raise was driven through the ore a distance of 90 ft. (27.5 m.). The general line of development was carried on from these prospect drifts after the mine was purchased. The east drift was extended northeast under open pit and an incline raise connection was made for handling this ore through the mine to the mill. From the raise off the north crosscut a drift was driven due east on the hanging wall of the upper run of ore and stopes opened up at intervals along this drift, the same heading and bench system of mining worked as at No. 1, using the sublevel tram system for handling the ore. The east drift on

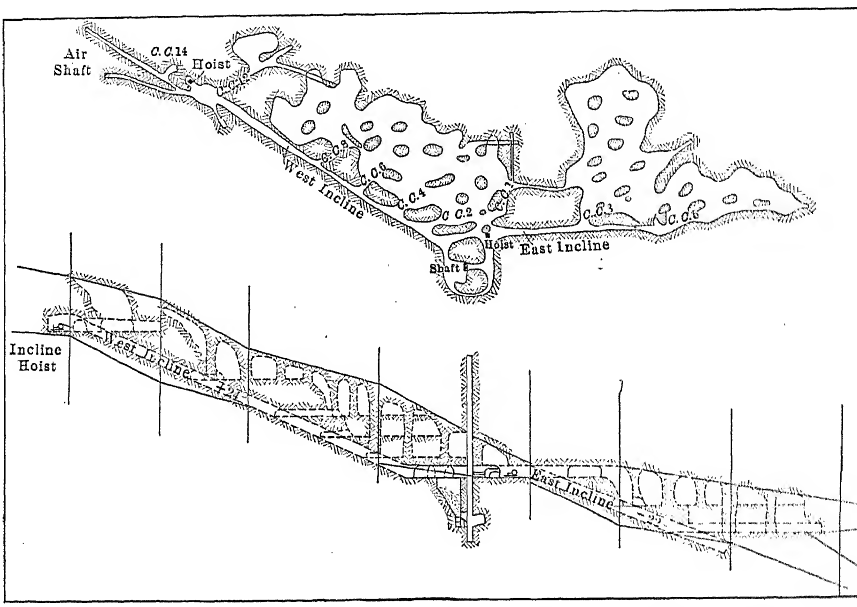



Fig. 5.—Plan and section showing incline system of mining at No. 2 mine.

the main level being driven in the middle of the ore, stopes were opened up on both the north and south side, and after cutting out the sill floor the overhead system of stoping was worked. In developing the ore below the main level, a 27° incline was driven a distance of 200 ft. following the foot wall. From this incline a drift was cut on west side approximately 95 ft. down. At the foot of the incline a foot-wall drift was driven west, which will eventually tap an orebody of 75,000 tons recently drilled out approximately 800 ft. west of this mine. On the east side, at foot of incline, a crosscut was driven to the hanging wall. This hanging wall was then followed a short distance and stopes opened up on the north side.

*Ore Breaking*

The drilling practice and equipment is the same throughout all the properties. Hammer drills are being used throughout. For heading and drifting work the 18-A Leyner and DR-6 Sullivan mounted drills are used.

For bench work, employing vertical holes, the Leyner machine taken off the shell and with improvised handles bolted to the backhead is used chiefly. The Dreadnaught machine is used for some classes of work. Some stope work is done with the Leyner machines on tripods, where long flat holes are required.

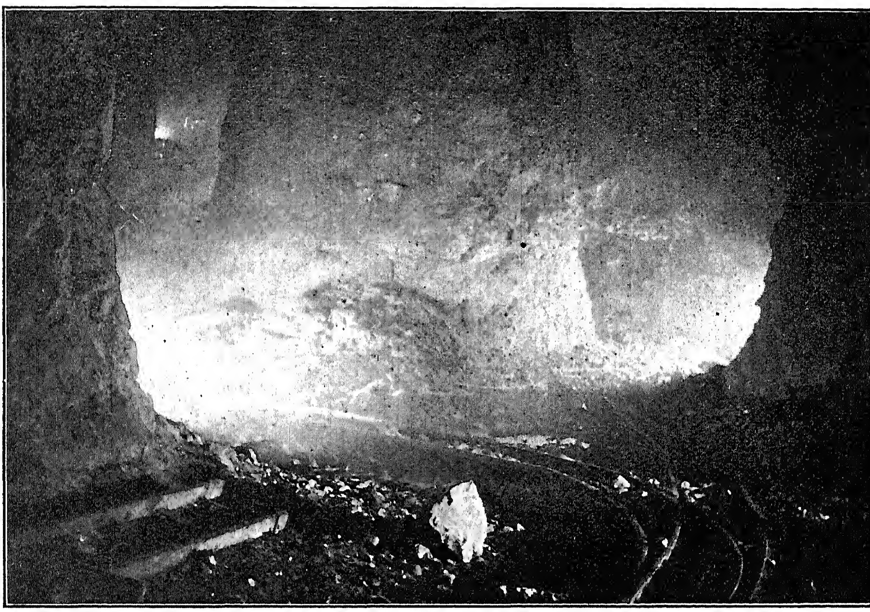


Fig. 6.—Pillar work in No. 1 mine.

This combination of drills for different classes of work is very desirable on account of using one kind of drill steel entirely, thus simplifying the question of sharpening.

The drill steel is 1¼ in. (31.75 mm.) (hollow round) with Leyner shank and Carr bit. The starters are made 2¼-in. (57.15-mm.) gage with 24-in. (60.96-mm.) changes and ⅛ in. (3.175 mm.) is allowed in gage on each change. Runs are made to 14 ft. (4 m.), but holes are rarely drilled deeper than 10 ft. (3 m.).

Leyner-Ingersoll drill sharpeners are used in all the shops. Both No. 3 and No. 5 models are in operation. The shops at mines No. 1 and No.

2 are underground and oil forges have been put into service. The shop at mine No. 3 is on the surface. All steel-sharpening shops are equipped with adequate slack pots, oil-tempering tanks, emery wheels for dressing shanks and individual steel rocks for each working place, from which the steel is checked out and in each shift.

For driving raises the Ingersoll-Rand BC-21 stoper is employed, using 1-in. (25.4-mm.) cruciform steel with the cross bit.

For shaft sinking, winze work, light stoping and blocking, the Ingersoll-Rand BCR-43 jackhamer is used with 7/8-in. (22-mm.) hollow hexagon steel. This steel has a collar shank. Both cross bits and Carr bits are used.

Fig. 7.—Main-level haulage way, No. 1 mine.

An air pressure of 80 lb. is maintained at the drills and water is supplied for wet drilling in order to keep down dust.

The ore breaking is all done on company time and each round is personally inspected by a shift boss to determine grade of ore, condition of roof, position of pillars, etc.

The development work, including drifts, crosscuts, raises, winzes, inclines and shafts, is commonly done on contract. These contracts are let on basis of feet advanced and contractor furnishes labor and explosives.

The loading is done on company time on a task basis of so many cars per man depending on conditions in each working place.

*Blasting*

Forty per cent. gelatin powder in $1\frac{1}{4}$ by 8-in. sticks is used for all blasting. Exploders are made up with Crescent fuse and No. 6 caps.

Electric detonators have been tried but the cap-and-fuse method has been found more desirable because a majority of holes are dry.

A separate crew of powdermen supervises the stope blasting, which is all done at the end of the night shift.

Underground magazines are provided to accommodate a ton or more of powder, but only one day's supply, or approximately half that amount, is kept on hand.

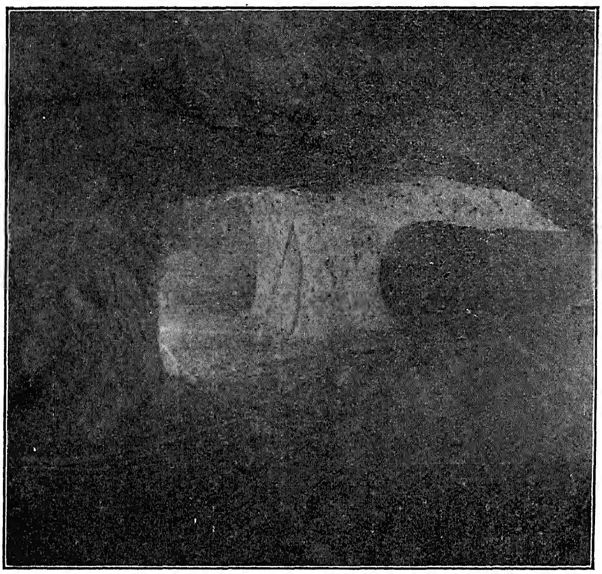


Fig. 8.—A stope in No. 2 mine.

*Tramming*

The sublevel tramming is done by hand with $1\frac{1}{4}$-ton end-dump cars. Two men load a certain task and tram each car to a raise. A maximum tram of 300 ft. is maintained.

The main-level haulage is done with 2-ton cars handled by 3-ton storage-battery locomotives. These locomotives travel from 4 to 6 miles per hour and handle 5- to 8-car trips to the tipple.

The tipples in use are of the drum type, motor-driven. One or two cars are run into the tipple and are dumped by being turned a complete revolution about the long axis.

The ore is dumped into a skip pocket having a capacity of 600 tons, and is drawn out through gates operated pneumatically, into 4-ton skips

working in balance. The skips dump automatically into a feeder above the gyratory crusher.

The man cages are double-decked and inclosed with chains or telescope gates. Counterweights are provided so that the cages also work in balance.

All cage and skip hoists are motor-driven and connected with herringbone gears.

### *Mine Drainage and Pumping*

Due to the fact that the orebody lies very close to the Holston River, and in places underneath creeks tributary to it, the question of water is of greatest importance. No. 1 mine has been flooded on two occasions and No. 2 mine had two lower levels under water at one time. This question in itself could be made the subject of a paper, but it is sufficient to say that pumps are installed at the three mines sufficient to handle 17,000 gal. per minute and pumping plants are under construction for an additional 10,000 gal. per minute.

None of the mines are directly connected underground at this time, but because of fissures and caves in the rock, the water during flood times drains from one to the other, making it necessary to have concrete bulkheads and very large pump installations.

The high-speed centrifugal pump is largely used, although plunger pumps of the horizontal duplex and vertical triplex type comprise part of the equipment. All the pumps are motor-driven, either direct- or gear-connected.

## Power

Electrical power is obtained from the Tennessee Power Co.'s hydroelectric plant at Parksville. This plant is located in Polk County 100 miles from Mascot, and a 66,000-volt transmission line connects the two places.

The Zinc company has an emergency power plant consisting of a 1000-hp. steam turbine sufficient to carry the pumping load at any time the power is out of commission.

The power plant for compressed air includes two Ingersoll-Rand 1600-cu. ft. per minute compressors and one Nordberg 3500-cu. ft. per minute compressor, all being direct-connected to synchronous motors.

## Labor

Table 1 shows distribution of the labor for a mine producing 1300 to 1350 tons per day.

All loading and tramming is done by negro labor; drilling and miscellaneous work by native white labor.

TABLE 1.—*Distribution of Labor*

| | Day Shift | Night Shift |
|---|---|---|
| Shift boss | 1 | 1 |
| Shoveler bosses | 2 | 2 |
| Machine men | 18 | 16 |
| Machine helpers | 18 | 16 |
| Blacksmiths | 2 | 2 |
| Shovelers | 44 | 42 |
| Chute pullers | 6 | 6 |
| Powdermen | .. | 4 |
| Pipemen | 4 | .. |
| Roofmen | 5 | .. |
| Timbermen | 2 | .. |
| Car repairmen | 2 | .. |
| Tool boys | 1 | 1 |
| Cage men | 1 | 1 |
| Trackmen | 4 | 2 |
| Motormen | 2 | 2 |
| Motor couplers | 2 | 2 |
| Incline hoistermen | 2 | 2 |
| Incline couplers | 4 | 4 |
| Tipplemen | 1 | 1 |
| Skip loaders | 2 | 2 |
| Total | 123 | 106 |

On a tons-per-man basis, the following is a fair example of results obtained: 15.7 tons per shoveler; 5.7 tons per man (total underground).

The pumping crew is not included in any of the above figures.

TABLE 2.—*Costs*

| | Per Ton |
|---|---|
| Drilling and boulder breaking | $0.220 |
| Blasting | 0.110 |
| Roof protection | 0.015 |
| Loading | 0.150 |
| Haulage and car repair | 0.080 |
| Hoisting and skip loading | 0.040 |
| Tracking | 0.015 |
| Foremen and miscellaneous | 0.070 |
| | $0.700 |

This includes about 40 ft. of development per week, which represents $0.04 of the breaking cost. Pumping expense is quite variable, but under normal conditions amounts to $0.03 per ton of ore handled and is not included in the above.

# Resistance of Artificial Mine-roof Supports*

BY WILLIAM GRIFFITH,† C. E., SCRANTON, PA.

(St. Louis Meeting, October, 1917)

THE purpose of this paper is to make public record of new information in regard to the sustaining power of artificial mine-roof supports (not timber props), the result of investigations recently made in the anthracite coal fields of Pennsylvania.

*First.*—By the "Scranton Mine Cave Commission," appointed to investigate the mining conditions under the City of Scranton.

*Second.*—By the "Pennsylvania Mine Cave Commission," appointed by the Governor to investigate the general subject of mine caves in the anthracite region of Pennsylvania.

*Third.*—By the personal researches and tests made by the author in an effort to secure a better artificial mine-roof or surface support.

During the progress of the work of the Scranton Mine Cave Commission, the Engineers for the Commission, Messrs. Conner and Griffith, being aware of the general lack of exact information in the engineering profession as to the strength of the ordinary artificial mine-roof or surface support, and particularly the stronger kinds, such as mine cogs, rock piers, etc., had tests made at the Fritz Engineering Laboratory at Lehigh University, to determine the sustaining power under various compressions of the several kinds of artificial mine supports in use in the anthracite region of Pennsylvania. A report of these tests was included in the report of this Commission, which was published as *Bulletin* No. 25 of the United States Bureau of Mines, Washington.

A similar series of tests, more extensive and more elaborately planned, were subsequently made for the Pennsylvania Mine Cave Commission at the U. S. Government Testing Laboratory at Pittsburgh, Pa. The report of this Commission was made to the Governor, but has not yet been published.

The author, during his connection with the Scranton Mine Cave Commission, observed that of all the various devices employed for sustaining the roof of coal mines, the one universally used and the best

* Read before the Pennsylvania Anthracite Section.

† Consulting Mining Engineer and Geologist.

known—the "timber cog," consisting of a cribwork of logs filled with mine rock and rubbish—was possessed of comparatively small resistance under the initial pressure, but withal was exceedingly elastic, its sustaining power increasing rapidly under compression. Such cogs or cribs would not fail completely until the compression amounted to one-third or one-half the original height of the structure.

The kind of roof support that seemed to have the greatest initial resistance was found to be concrete piers, which, although somewhat

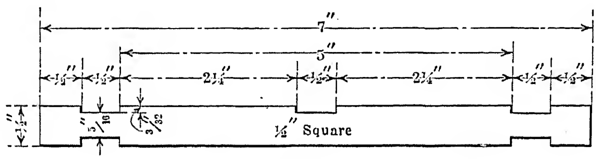



FIG. 1.

costly, are nevertheless used to a small extent in some portions of the anthracite region; but the tests showed that such rigid piers would fail completely when the load upon them was sufficient to cause a compression of about 3 per cent. of the total height, and that this failure of concrete piers under pressure was sudden and without warning, thus being a menace to the safety of the miners.

It was desirable, therefore, if possible, to devise some sort of artificial roof support which would partake of both the elasticity of the timber cog

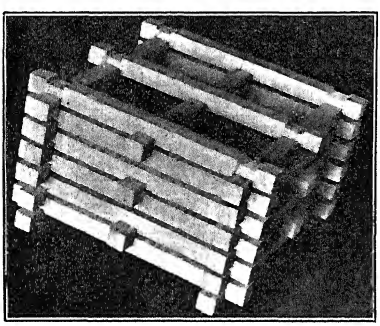


FIG. 2.

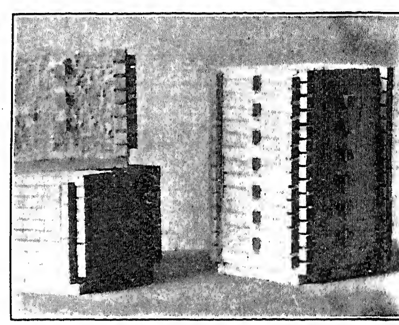


FIG. 3.

and the rigidity of the concrete pier, and the author tested a number of devices in an effort to accomplish this end. The result of this series of tests was the production of what is referred to in this paper as "Griffith's Mine Pier," which consists essentially of a timber crib, each element or member of which is provided with a series of notches as shown in Fig. 1, and which may be framed in quantity, by machinery, and creosoted outside of the mine, so that the crib may be easily erected inside, the members fitting together one upon the other and forming a rigid cribwork similar to

that shown in Fig. 2. Concrete is poured into this crib, thus forming an artificial mine pier of simple construction, yet having an initial resistance equal to a concrete pier of the same dimensions, but which is so elastic that it will not fail under pressure until the total compression amounts to nearly 15 per cent. of the original height and the ultimate load sus-

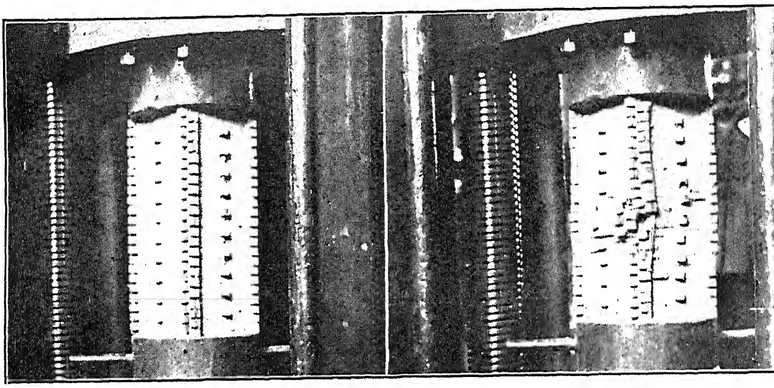
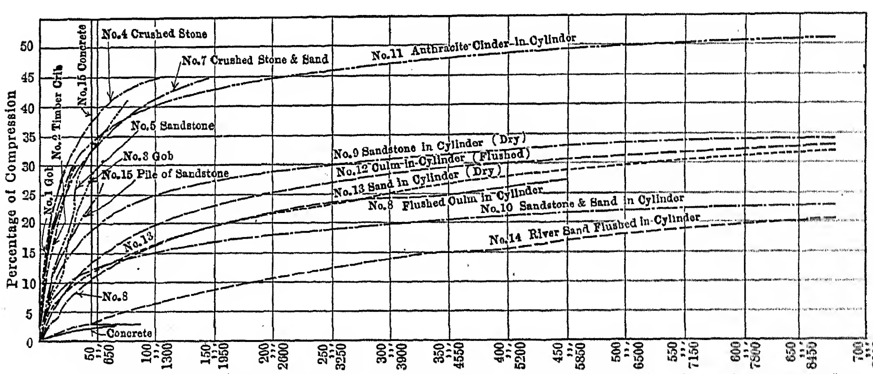


FIG. 4. FIG. 5.

FIGS. 4 AND 5.—SPECIMEN J BEFORE AND AFTER TEST.

tained is equal to about three times the ultimate strength of concrete piers of the same dimensions, and about 20 times the resistance of the well-built timber cog. We have thus produced a concrete pier reinforced with notched timbers, in the manner described.




For Descriptions of Above Tests see Table 1

FIG. 6.—TESTS OF SCRANTON MINE-CAVE COMMISSION AT LEHIGH UNIVERSITY.

The variety of concrete proved by the tests to be most efficient seems to be the cheapest mixture that can be made, viz., cyclopean concrete composed of pieces of mine rock as large as can be conveniently handled by one or two men; the interstices between these pieces of rock being filled with ordinary concrete grouting composed of cement, sand and small

broken stone, the object being to secure the greatest density possible. In preparing the crib mentioned, the notches in the timber should be of sufficient depth—that is, nearly one-fourth the thickness of the timber—

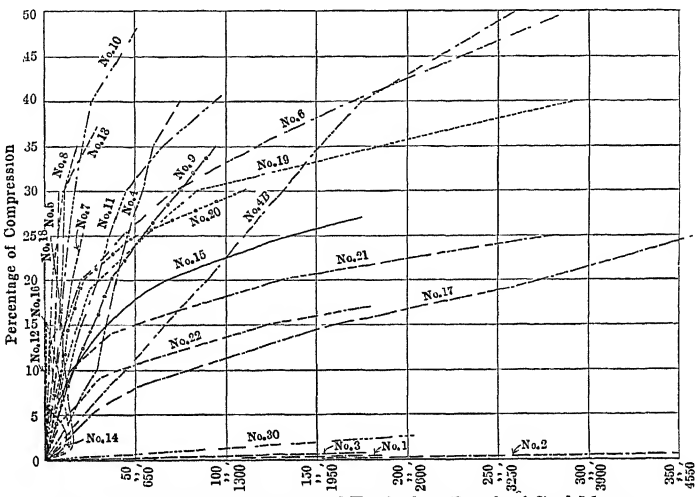



FIG. 7.—TESTS FOR THE PENNSYLVANIA STATE MINE-CAVE COMMISSION AT GOVERNMENT LABORATORY, PITTSBURGH, PA.

so that when the crib is finished the timbers will be separated by a space of about 1 in. or less. This permits the grout to flow in and partly fill the space between the timbers during the course of construction. After the

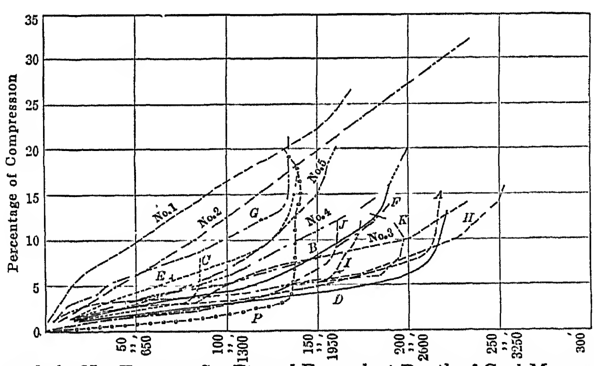



FIG. 8.—TESTS OF GRIFFITH'S MINE PIER AT D. L. AND W. LABORATORY, SCRANTON, PA.

structure is completed the unfilled spaces may be pointed with a trowel, from the outside, after which the outside of the pier may be coated with cement by the use of a cement gun or any other device.

It will be impracticable to build such a crib tight against the mine roof. There will be a small space which should be filled by ramming with

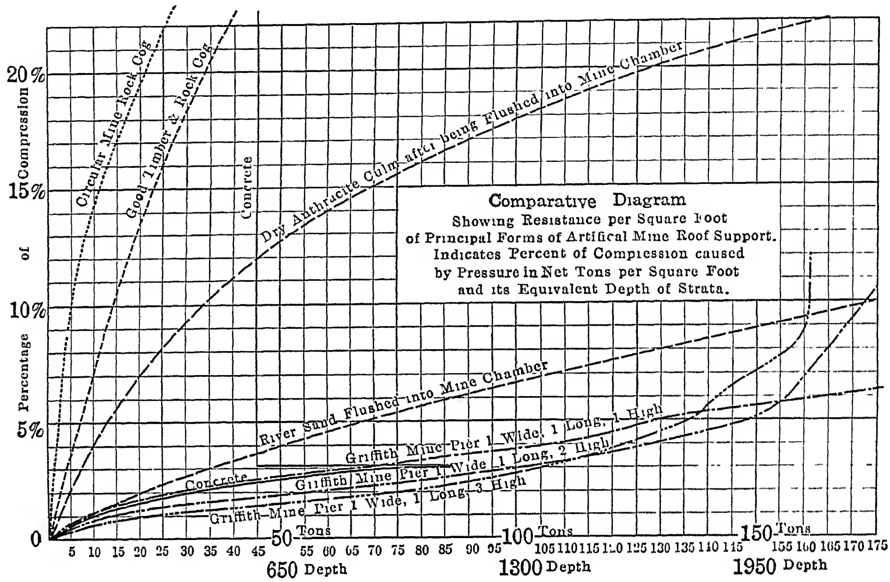



FIG. 9.

dry concrete; that is, concrete mixed with a small proportion of water. Thus will be formed a very lasting mine pier, because each timber will be

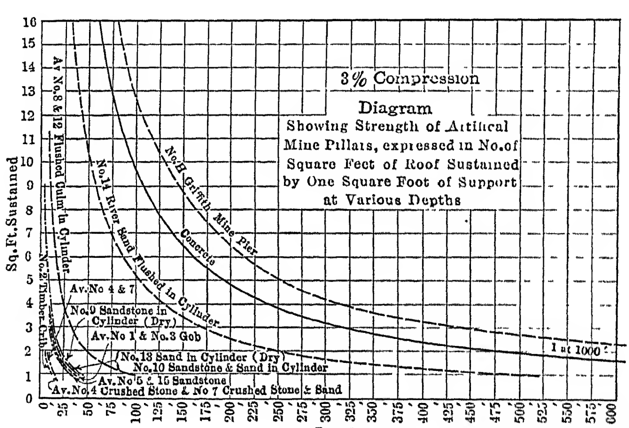



FIG. 10.

surrounded or embedded in concrete, and if the timbers are first creosoted the construction should be very durable.

Fig. 3 shows test specimens *H, I,* and *K* as they appeared imme-

diately after completion. Figs. 4 and 5 show the condition of specimen *J* before and after the load had been applied.

In nearly all cases, it was found that there was an elasticity of about 3 per cent. in the pier when the load was removed. In other words, the

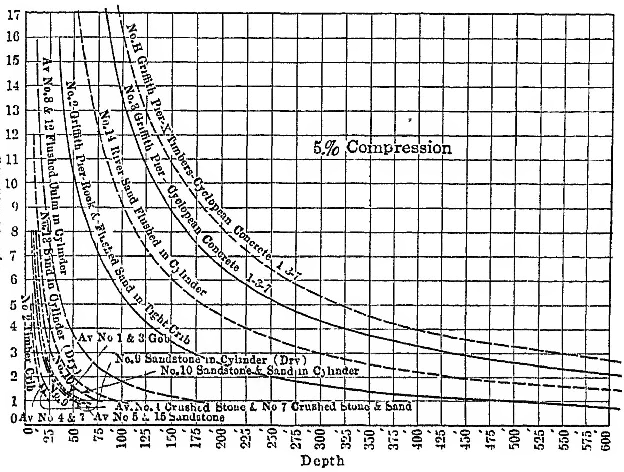



FIG. 11.

height of a tested piece, after the load was removed, was about 3 per cent. more than the height under greatest pressure. Some of the piers were

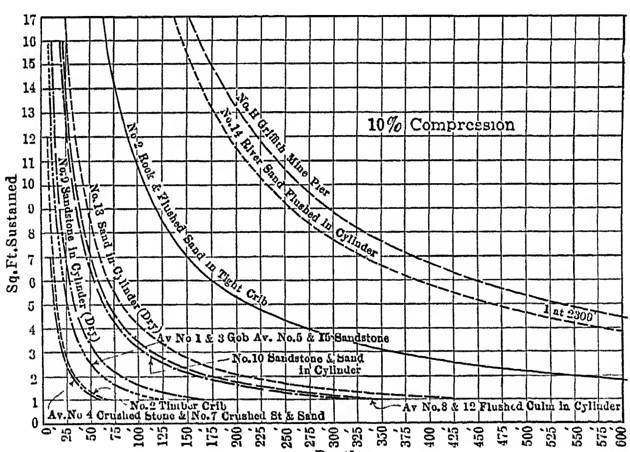



FIG. 12.

tested the second time and sustained the same weight, under the same compression, as in the first test.

There are many localities and circumstances in coal and metal mines where strong, elastic roof supports of this sort would be exceedingly

useful, and the cost of placing the same for such emergencies would not be excessive.

Since it is a practical impossibility to introduce artificial roof support in a mine in such a manner as to prevent minute subsidence, it is necessary to assume a certain amount of surface settlement as permissible. The following tabulations and preceding diagrams show the approximate sustaining power of the various kinds of artificial roof support which may be used in mines, and the percentage of compression that will be produced by the load sustained:

Table 1 and Fig. 6, tests made at Lehigh University for the Scranton Mine Cave Commission.

Table 2 and Fig. 7, tests made at the U. S. Government Laboratory for the Pennsylvania State Mine Cave Commission.

Table 3 and Fig 8, tests made at the D. L. & W. Laboratory, Scranton, of Griffith's Mine Pier.

The figures at the head of each vertical column indicate the per cent. of compression and the figures in the body of the tabulation are the loads sustained per square foot of horizontal area of the support in net tons.

TABLE 1.—*Resistance per Square Foot of Mine-roof Supports*

Tests for the Scranton Mine Cave Commission at Lehigh University

| Test No. | Description of Test | Load in Net Tons for Various Percentages of Compression | | | | | | | | | | | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 1 % | 2 % | 3 % | 4 % | 5 % | 6 % | 7 % | 8 % | 9 % | 10 % | 11 % | |
| 2 Tests 1 and 3.. | Pile of mine gob.......................... | ½ | .... | .... | ..... | 2–7 | ..... | ..... | ..... | ....... | 3–15 | ....... | Free to expand laterally. |
| 2....... | Timber and rock cog, average construction....... | ½ | .... | .... | ..... | 2½ | ... | ..... | ..... | ....... | 5 | ....... | Free to expand laterally, not notched. |
| 4....... | Pile of crushed stone........................ | ½ | .... | .... | ..... | 2½ | ..... | ..... | ..... | ....... | 4 | ....... | Free to expand laterally. |
| 7....... | Pile of crushed stone and sand.................. | ½ | .... | .... | ..... | 2½ | ..... | ..... | ..... | ....... | 4 | ....... | |
| 2 Tests 5 and 15. | Pile broken sand rock, size cocoanut and larger... | ... | 2 | .... | ..... | 4 | ..... | ..... | ..... | ....... | 9½ | | |
| 2 Tests 12 and 8. | Anthracite culm, flushed (dried)................. | 2½ | 5 | 6–8 | 8–11 | 10–15 | 14–20 | 17–25 | 20–30 | 25–35 | 27–40 | 32–47 | Confined, no lateral expansion. Dry when tested. |
| 9....... | Crushed sandstone, 40 per cent. voids .......... | 1 | 2 | 3 | 4 | 5 | 6½ | 8 | 10 | 11½ | 12½ | 14 | Confined, no lateral expansion |
| 10...... | Crushed sandstone, voids filled with sand ....... | 1 | 2 | 3½ | 5 | 7 | 9 | 11 | 13 | 18 | 25 | | |
| 11...... | Anthracite coal ashes, flushed.................. | 1 | .... | .... | ..... | 2 | ..... | ..... | ..... | ....... | 4 | ....... | |
| 13...... | Dry sand, filling flushed .... .................. | 1 | 2 | 3 | 4 | 7 | 10 | 15 | 20 | 25 | 31 | 39 | |
| 14...... | Sand, filling flushed ......................... | 7 | 20 | 40 | 55 | 70 | 90 | 110 | 132 | 155 | 180 | 205 | |
| 15...... | Concrete block, age 6 months.................. | 10 | 30 | 80 | 45 | 45 | 40 | | 35 | 35 | 30 | 25 | |

TABLE 1 (*Continued*).—*Resistance per Square Foot of Mine-roof Supports*

Tests for the Scranton Mine Cave Commission at Lehigh University

| Test No. | Description of Test | Load in Net Tons for Various Percentages of Compression | | | | | | | | | | | | | | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 12% | 13% | 14% | 15% | 16% | 17% | 18% | 19% | 20% | 25% | 30% | 35% | 40% | 50% | |
| 2 Tests 1 and 3.. | Pile of mine gob | ... | | .. | 5–27 | ... | .. | . | ..... | 9–30 | 15–44 | 25–58 | ... | .. | ... | Free to expand laterally. |
| 2....... | Timber and rock cog, average construction | .... | . | . | 10 | .... | .. | .. | .. | 15 | 22½ | 32 | .... | | . | Free to expand laterally. Not notched. |
| 4....... | Pile of crushed stone | ... | . | . | 6½ | ... | | | . | 9½ | 14 | 23 | 45 | 63 | | Free to expand laterally. |
| 7....... | Pile of crushed stone and sand | ...... | .. | ... | 7 | .. | . | . | . | 14 | 22½ | 41 | 57 | 90 | | |
| 2 Tests 5 and 15. | Pile broken sand rock, size cocoanut and larger | ...... | .. | ...... | 15–20 | ... | . | . | ... | 26–34 | 30–50 | 39–63 | .. | ... | . | |
| 2 Tests 12 and 8. | Anthracite culm, flushed (dried) | 37–55 | 42–64 | 48–75 | 55–85 | 62–95 | 71–105 | 80–120 | 90–137 | 102–157 | 187–360 | 400–600 | . | .. | | Confined, no lateral expansion Dry when tested. |
| 9....... | Crushed sand stone, 40 per cent voids | 17 | 20 | 24 | 27 | 31 | 35 | 40 | 47 | 52 | 100 | 255 | 34% =650 | ..... | .... | Confined, no lateral expansion. |
| 10...... | Crushed sand stone, voids filled with sand | 42 | 55 | 75 | 95 | 115 | 150 | 195 | 250 | 315 | 23% =360 | .. | . | . | | |
| 11...... | Anthracite coal ashes, flushed | .... | .... | ... | 7 | .... | .. | . | .. | 12 | 19 | 30 | 50 | 96 | 525 | |
| 13...... | Dry sand, filling flushed | 47 | 56 | 67 | 80 | 90 | 105 | 120 | 137 | 157 | 295 | 520 | 32% =660 | | | |
| 14...... | Sand, filling flushed | 237 | 268 | 300 | 345 | 410 | 445 | 500 | 555 | 635 | .. | . | . | | . | |
| 15...... | Concrete block, age 6 months.. | . | . | ... | .. | .. | . | | | | | .. | | | | |

TABLE 2.—*Resistance per Square Foot of Mine-roof Supports*

Tests for the Penna. State Mine Cave Commission at Govt. Laboratory, Pittsburgh

| Test No. | Description of Test | Compression, Tons | | Load in Net Tons for Various Percentages of Compression | | | | | | | | | | | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | 1% | 2% | 3% | 4% | 5% | 6% | 7% | 8% | 9% | 10% | 11% | |
| 1 | Mine prop, yellow pine | 5/10 of 1% | 190 | | | | | | | | | | | | |
| 2 | Mine prop, chestnut oak | 8/10 of 1% | 395 | | | | | | | | | | | | |
| 3 | Group of mine props, yellow pine | 6/10 of 1% | 176 | | | | | | | | | | | | |
| 4 | Mine timber transversely crush, approx. for yellow pine | | | | | 8 | | 15 | | | 22½ | | 28 | | |
| 4-B | Mine timber transversely crush, approx. for chestnut oak. | | | | | | | 22 | | | 35 | | 45 | | |
| 5 | Mine-rock cog, rubble shell, rubbish center | | | | | | | 2 | | | | | 4½ | | |
| 6 | Circular mine rock cog | | | | | | | 2 | | | | | 5 | | |
| 7 | Yellow pine timber rock cog | | | 1 | 2 | 2½ | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 9½ | |
| 8 | Yellow pine timber cog, no filling | | | 1 | | | | 2 | | 2½ | | | 4 | | |
| 9 | Oak timber and rock cog | | | 1 | 2 | 3 | 4 | 6 | 7½ | 9 | 12 | 14 | 16 | 18 | |
| 10 | Oak timber cog, no filling | | | 1 | 1½ | 2 | 2½ | 3 | 3½ | 4 | 4 2 | 5 | 6 | 6½ | |
| 11 | Spruce timber and rock cog | | | 1½ | 3 | 4 | 5½ | 7½ | 8 | 9 | 10½ | 11½ | 12½ | 14 | |
| 12 | Mine-rock cog, loose, voids | | | | | | | 3/10 | | | 6/10 | | | | |
| 13 | Mine-rock cog, voids filled with sand and ashes | | | | | | | 1 | | | | | 2 | | |
| 14 | Square rock and cement shell cog flushed culm center | | | 7½ | 11 | 11 | 9 | 6½ | | | | | | | |
| 15 | Flushed culm (damp)? | | | 1 | 2 | 3 | 5 | 6½ | 7½ | 9 | 10 | 12½ | 15 | 18 | Confined, damp when tested. |
| 16 | Masonry shell, center flushed with sand | | | 10 | 14 | 13 | 12½ | 12 | 11 | 10 | 9 | 8 | 7½ | 7 | |
| 17 | Flushed sand in cylinder. | | | 5 | 10 | 15 | 20 | 25 | 32½ | 42 | 50 | 65 | 78 | 95 | Confined, not free to expand laterally |
| 18 | Mine-rock shell, green mortar filled with sand | | | 12 | 16 | 15 | 14 | 13½ | 13 | 12½ | | | 11½ | | |
| 19 | Ashes flushed in cylinder | | | | | | | 5 | | | | | 10 | | Confined, not free to expand laterally |
| 20 | Broken mine refuse in cylinder | | | | | | | 3 | | | | | 8 | | |
| 21 | Sand and ashes in cylinder. | | | | | | | 6 | | | | | 13 | | |
| 22 | Crushed mine rock and sand in cylinder | | | | | | 5 | 8 | | | | 30 | 40 | | |
| 30 | Concrete column 1–2–4. 25d-steel reinforcement | | | 75 | 150 | 2⅔%=200 | | | | | | | | | |

TABLE 2 (*Continued*).—*Resistance per Square Foot of Mine-roof Supports*
Tests for the Penna. State Mine Cave Commission at Govt. Laboratory, Pittsburgh

| Test No. | Description of Test | Load in Net Tons for Various Percentages of Compression | | | | | | | | | | | | | | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 12% | 13% | 14% | 15% | 16% | 17% | 18% | 19% | 20% | 25% | 30% | 35% | 40% | 50% | |
| 1 | Mine prop, yellow pine. | | | | | | | | | | | | | | | |
| 2 | Mine prop, chestnut oak. | | | | | | | | | | | | | | | |
| 3 | Group of mine props, yellow pine. | | | | | | | | | | | | | | | |
| 4 | Mine timber transversely crush, approx. for yellow pine.. | .... . | .. | ..... | 34 | ...... | ... | ... | ..... | 40 | 46 | 54 | 60 | 75 | | |
| 4-B | Mine timber transversely crush, approx. for chestnut oak | ..... | . .. | .... | 67 | ...... | ... | 81 | ..... | 90 | 110 | 132 | 152 | 175 | 260 | |
| 5 | Mine-rock cog, rubble shell, rubbish center.......... | ...... | .... | ...... | 5 | ...... | ... | .. | ..... | 6 | 7 | 7½ | | | | |
| 6 | Circular mine rock cog.......... | . . | . | . .. . | 11 | ...... | ... | ..... | ...... | 22 | 43 | 72 | 117 | 170 | 290 | |
| 7 | Yellow pine timber rock cog.......... | 10 | 10½ | 11 | 12 | ...... | 12½ | .. | ..... | 15 | 21 | | | | | |
| 8 | Yellow pine timber cog, no filling.......... | ... .. | 5½ | .... | 7 | ...... | . | .. | ... .. | 8½ | 10 | 11½ | 18 | | | |
| 9 | Oak timber and rock cog.......... | 20 | 22½ | 24 | 27 | 28½ | 31 | 33 | 35 | 37 | 54 | 75 | 95 | | | |
| 10 | Oak timber cog, no filling.......... | 7 | .... | 7½ | 8 | 8½ | 9 | 9½ | 10 | 11 | 13 | 17 | 22 | 26 | 48% =52 | |
| 11 | Spruce timber and rock cog.......... | 15½ | 17 | 18 | 20½ | .... | ... | ... | ...... | 29 | 36 | 45 | 65 | 95 | | |
| 12 | Mine-rock cog, loose, voids.......... | ... .. | .... | ...... | 9⁄10 | | | | | | | | | | | |
| 13 | Mine-rock cog, voids filled with sand and ashes.......... | ...... | .... | ...... | 2½ | .... | ... | .... | ... .. | 3½ | 6 | 10 | 23 | 37% =30 | | |
| 14 | Square rock and cement shell cog flushed culm center. | | | | | | | | | | | | | | | |
| 15 | Flushed culm (damp)?.......... | 22 | 25 | 30 | 34 | 40 | 45 | 52 | 60 | 68 | 137 | 27% =175 | ... | .... | ... | Confined, damp when tested. |
| 16 | Masonry shell, center flushed with sand.......... | 6 | 4 | 3 | 2 | 1 6 | | | | | | | | | | |
| 17 | Flushed sand in cylinder.......... | 110 | 127 | 142 | 158 | 183 | 207 | 230 | 255 | 273 | 360 | .... | .... | .... | ... | Confined, not free to expand laterally. |
| 18 | Mine rock shell, green mortar filled with sand.......... | ..... | .... | ...... | 9½ | ...... | ... | ..... | .. . | 6 | 3 | | | | | |
| 19 | Ashes flushed in cy inder.......... | ..... | .... | ..... | 18 | .... . | ... | .. | 25 | 30 | 55 | 85 | 185 | 290 | .. . | Confined, not free to expand laterally |
| 20 | Broken mine refuse in cylinder.......... | ..... | .... | ...... | 13 | ..... | .. | .. | .... | 22 | 50 | 110 | ... | .. | | |
| 21 | Sand and ashes in cylinder.......... | .. ... | .... | 37 | 48 | ... .. | . | ... | .... . | 130 | 280 | | .. | . | ... | |
| 22 | Crushed mine rock and sand in cylinder.......... | .. ... | .... | ...... | 123 | ..... | 180 | ... | . .. | | .. | ... | . | . | | |
| 30 | Concrete column 1–2–4 25d-steel reinforcement. | | | | | | | | | | | | | | | |

TABLE 3.—*Resistance per Square Foot of Mine-roof Supports*

Tests of Griffith's Mine Pier at D. L. & W. Laboratory, Scranton

| Test No. | | Description of Test | Load in Net Tons for Various Percentages of Compression 1% | 2% | 3% | 4% | 5% | 6% | 7% | 8% | 9% | 10% | 11% | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Size of test Age 22 days | 1 | Tight crib flushed with sand (dried)........... | 2½ | 5 | 8 | 12 | 15 | 20 | 26 | 41 | 45 | 51 | 58 | Plan of cribs A to G |
| 8″ cribs | 2 | Open crib notched ⅛″, large rock and sand fill . | 7½ | 15 | 25 | 33 | 45 | 50 | 58 | 65 | 75 | 82 | 90 | as shown in Fig. 2, |
| 6″ center | 3 | Open crib notched ⅛″, large rock, concrete 1–3–7 | 10 | 30 | 80 | 90 | 100 | 110 | 125 | 150 | 177 | 200 | 208 | without center cross- |
| 1″ timber | 4 | Open crib notched ⅛″, small rock, concrete 1–3–7 | 10 | 25 | 45 | 65 | 85 | 100 | 110 | 120 | 130 | 140 | 150 | pieces. |
| | 5 | Open crib not notched, small rock, concrete 1–3–7 | 10 | 25 | 45 | 65 | 78 | 95 | 102 | 110 | 119 | 126 | 130 | |
| | A | Tight crib large rock voids filled, concrete 1–3–2 small stone.............................. | 10 | 30 | 80 | 125 | 155 | 173 | 190 | 207 | 213 | 215 | 216 | |
| | B | Tight crib large rock mix 1–3–2 rice stone, 8 parts 1″ stone............................ | 10 | 30 | 60 | 90 | 115 | 127 | 140 | 147 | 155 | 165 | 172 | |
| 6″ cribs | C | Concrete block Same mixture as B ......... | 10 | 45 | 75 | 82 | 84 | 85 | 85 | 85 | | | | |
| 5″ center | D | Tight crib 1″ and 2″ stone grouted with 1–3 mortar. ................................ | 18 | 60 | 105 | 145 | 180 | 197 | 208 | 214 | 216 | 217½ | 218½ | |
| ½″ timber | E | Tight crib concrete 1–3–4, buckwheat size stone.. | 10 | 20 | 33 | 45 | 62½ | 75 | 90 | 105 | 118 | 125 | 128 | |
| | F | Tight crib concrete 1–6 run of crusher. ......... | 10 | 25 | 45 | 60 | 100 | 115 | 133 | 148 | 160 | 167 | 175 | |
| Age 35 days | G | Tight crib large and small rock sand flushing ... | 5 | 8 | 15 | 28 | 35 | 47 | 60 | 75 | 89 | 98 | 108 | |
| | H | Crib notched 3/32″–⅛″ open X timbers, concrete like A . ....... ....................... | 7 | 28 | 61 | 111 | 130 | 170 | .... | 199 4 | ..... | 226 | ....... | Plan of cribs H to J |
| | I | Crib, same as H 10″ high.................... | 11 | 53 | 104 | 129 | 149 | 159 | .... | 164 | ..... | 172 | ....... | as shown in Fig. 2. |
| | J | Crib, same as H 15″ high.................... | 24 | 78 | 105 | 123 | 136 | 142 | .... | 158 | ..... | 161 | . ... | |
| | K | Tight crib mine slate fill like D............. . | 3.2 | 10.2 | 28 | 104 | 136 | 183 | ..... | 195 | ..... | 197 | ..... | Crib same as A to G. |
| Age 42 days | P | Open crib notched ⅛″, large rock and concrete 1–3–7, 14″ crib, 12″ center, 2″ timber.. . | 73 | 116 | 130 | 134 | 135 | 136 | ..... | 137 | ..... | 138 | ...... | Tested at U S Govt. Laboratory, P'gh, Pa |

TABLE 3 (*Continued*).—*Resistance per Square Foot of Mine-roof Supports*

Tests of Griffith's Mine Pier at D. L. & W. Laboratory, Scranton

| Test No | | Description of Test | Load in Net Tons for Various Percentages of Compression | | | | | | | | | | | | | | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 12% | 13% | 14% | 15% | 16% | 17% | 18% | 19% | 20% | 25% | 30% | 35% | 40% | 50% | |
| Size of test | | | | | | | | | | | | | | | | | |
| Age 22 days | 1 | Tight crib flushed with sand (dried) | 65 | 73 | 80 | 90 | 97 | 105 | 115 | 125 | 133 | 163 | | . | | . | Plan of cribs A to G as shown in Fig 2, without center cross-pieces. |
| 8″ cribs | 2 | Open crib notched ⅛″, large rock and sand fill | 98 | 104 | 110 | 118 | 124 | 130 | 138 | 145 | 151 | 185 | 220 | | | | |
| 6″ center | 3 | Open crib notched ⅛″, large rock, concrete 1–3–7 | 215 | 228 | 232 | | | | | | | | | | | | |
| 1″ timber | 4 | Open crib notched ⅛″, small rock, concrete 1–3–7 | 160 | 170 | 178 | 185 | 190 | 192 | 195 | 197 | 200 | | | | | | |
| | 5 | Open crib *not* notched, small rock, concrete 1–3–7 | 135 | 140 | 145 | 150 | 152 | 154 | 155 | 157½ | 160 | | | | | | |
| | A | Tight crib large rock voids filled, concrete 1–3–2 small stone | 216½ | 217 | 217½ | | | | | | | | | | | | |
| | B | Tight crib large rock mix 1–3–2 rice stone, 8 parts 1″ stone | 179 | 184 | 187 | 188 | 189 | | | | | | | | | | |
| 6″ cribs | C | Concrete block Same mixture as B | | | | | | | | | | | | | | | |
| 5″ center | D | Tight crib 1″ and 2″ stone grouted with 1–3 mortar | 220 | 222 | | | | | | | | | | | | | |
| ½″ timber | E | Tight crib concrete 1–3–4, buckwheat size stone | 133 | 135 | 136½ | 137½ | 138½ | 139 | | | | | | | | | |
| | F | Tight crib concrete 1–6 run of crusher | 181 | 185 | 190 | | | | | | | | | | | | |
| Age 35 days | G | Tight crib large and small rock sand flushing | 117 | 128 | 132 | 134 | 134 | 134 | 134 | 134 | 134 | | | | | | |
| | H | Crib notched 3/32″–⅛″ open × timbers, concrete like A | ... | ... | 248 | ... | ... | | | ... | ... | ... | ... | ... | ... | ... | Plan of cribs H to J as shown in Fig 2 |
| | I | Crib, same as H 10″ high | 173.6 | | | | | | | | | | | | | | |
| | J | Crib, same as H 15″ high | 161 | | | | | | | | | | | | | | |
| | K | Tight crib mine slate fill like D | 191 | 175 | ... | ... | ... | . | ... | ... | .. | .. | ... | .. | .. | . | Crib same as A to G. |
| Age 42 days | P | Open crib notched ⅛″, large rock and concrete 1–3–7, 14″ crib, 12″ center, 2″ timber | 138 | ... | ... | 141 | ... | . | 138 | ... | 132 | .. | ... | ... | ... | ... | Tested at U. S. Govt. Laboratory, P'gh, Pa |

## Discussion

E. T. Conner, Scranton, Pa.—I had the honor of being associated with Mr. Griffith in an investigation of the Scranton mine cave problem several years ago, in the course of which we investigated rather carefully the bearing strength of various materials that might be utilized for the support of overburden. We found that there were no reliable data as to the resisting strength, for instance, of gob pillars or timber cogs or cribs, nor as to the amount of compression that might be expected under a given burden. We made a series of tests at the Fritz engineering laboratory of Lehigh University, from which we deduced certain conclusions, subject, however, to further test before they could be definitely recommended. Our study of the subject was later checked by a commission appointed by the Governor of the State of Pennsylvania to investigate the same subject, the subsidence of the surface and the danger entailed thereby. The State commission made a series of tests on a much more ambitious scale at Pittsburgh, in the laboratory of the U. S. Bureau of Mines, and their tests agreed quite closely with those that Mr. Griffith and I had made.

As the result of our study, we determined that the most elastic support is the ordinary timber cog or crib, while the most rigid support is a concrete pillar. We found, however, on account of the non-elasticity of a concrete pillar, that it would be unsuitable for such conditions as were encountered in the investigation of the mines under the city of Scranton, because any support that may be adopted must necessarily provide for some subsidence; this by reason of the fact that in that region the coal measures contain 11 beds of coal in a total depth of about 750 ft., the intervals between the several beds of coal varying considerably, and the character of the slates and shales also differing widely.

W. R. Crane, State College, Pa.—I have followed with considerable interest the development of Mr. Griffith's idea regarding the use of concrete cribs for mine support. As is well known, the crib is a universal means of support used in both coal and metal mines. It is filled ordinarily with waste rock, but rarely with foreign material introduced into the mine. Mr. Griffith has devised a support combining the essential features of the solid pier and the crib. He has filled the crib with concrete made of various-sized waste. Under pressure of a settling top, the concrete filling crushes, and the result is an ordinary crib filled with close-fitting but broken filling.

It is claimed that the concrete crib has considerable elasticity, but I can hardly see how that is of any practical value, seeing that the crib once under pressure continually remains in that condition until ultimately destroyed.

My chief objection to the acceptance of the facts given, as conclusive evidence of the action of large-sized supports, is that only models have

been used in the tests recorded and, further, that the testing was done under ideal conditions. The results of tests made in a testing machine are far different from those under actual mine condition, where the pressure is exerted by subsiding mine roof. Very rarely will the top settle uniformly over the full area of such a supporting crib, for the occurrence of cracks and slips will cause certain parts to move differently, producing a differential action that is very disastrous to any kind of a supporting surface. Should the top settle uniformly it may squeeze the crib between sloping surfaces, which would cause rapid disintegration.

It seems to me that we should have had in this paper the recorded results of some practical application of the support described, and I trust that some time in the near future we may have such data presented in a paper supplementing the present one.

H. M. Wilson, Pittsburgh, Pa.—The tests at the Bureau of Mines, to which Mr. Conner has referred, brought out very clearly certain indicative possibilities. I agree with Prof. Crane regarding the behavior of small test blocks filled with concrete. I had a good deal to do with the testing of concrete, in connection with the study of materials by the Bureau of Mines. Concrete has very little elasticity, while the confining crib is quite elastic. Necessarily the concrete must crush, and thereafter it acts like broken stone mixed with sand. The tests indicated that a properly filled crib is, so far as we yet know, the best supporting material for a mine. It will resist better than any other material the shearing stresses of which Prof. Crane has spoken, stresses which may come across a part of the crib only. We tried cubes of concrete, columns of reinforced concrete, concrete retained by steel bands of different dimensions, but for lack of money we had to stop long before we finished.

The tests showed not only that the properly filled crib is apparently the best supporting material, but that its strength depends largely on the nature of the filling material and the way it is placed in the crib. With big pieces of rock alone, the rock will be crushed and subside 50 or 60 per cent., but if you add the proper proportion of smaller stone, gravel, and sand, so as to fill all the interstices, you obtain practically the effect of a first-class concrete, the walls of the crib performing the same function as reinforcement in binding the aggregate together.

William Griffith (written discussion*).—We note that W. R. Crane remarks, "It is claimed that the concrete crib has considerable elasticity, but I can hardly see how that is of any practical value seeing that the crib once under pressure continually remains in that condition until ultimately destroyed." It may be worth while to explain more fully our views on the practical value of elasticity in a mine-roof support.

A mine-roof support, in our judgment, should be characterized by two essential features: elasticity, which conduces to safety; and resistance

* Received Jan. 24, 1918.

to compression, which affords the support. Supports of two types, massive, costly structures, and posts and small piers made entirely of concrete, have been used experimentally in our anthracite mines, but I believe the latter kind are now universally discarded because their lack of elasticity has proved dangerous. Such supports, when subjected to excessive pressure, do not subside or compress enough to be readily perceptible; nor do they show any other visible signs of stress until near the point of failure, and such failure is not only sudden but total. Failure *without warning* is a menace to the miners, as they are liable to be trapped in a squeeze or imprisoned behind a crushed area. With an elastic support, however, ample warning is given through the compression of the material; this is easily visible and may be measured from day to day by the mine foreman, who may thus judge whether it is advisable to abandon that portion of the mine before the final crush, or whether it would be possible to prevent further subsidence by installing supplementary supports.

Experience has taught that the most effective support is the one which, while sufficiently rigid, is elastic enough to permit the distribution of the roof weight over the largest possible area; this is the great advantage of the timber crib, which for years has been the ultimate resource for stopping a progressive squeeze or creep. In such an occurrence, a rigid, unyielding structure would individually sustain the whole load, or would suddenly crumble under the excessive weight; in the latter case, its crushed remains having no strength, the whole tremendous pressure would immediately be transferred to the next support, which would in turn be crushed. On the other hand, if such piers were elastic, the excessive weight would simply compress them until other supports in the vicinity could take their portions of the load, which would thus be distributed over a large number of elastic supports in each of which the resistance increases with compression. The combined resistance of a number of such elastic supports, each accommodating itself to inequalities of roof and floor, would perhaps bring about equilibrium, or cause cracking of the roof, thus relieving the strain and stopping the progress of the creep more effectively than rigid piers. It is the elasticity of a mine-roof support which enables it to cope with unequal settlement, and fit the irregularities of surface in roof and floor, referred to in the latter part of Dr. Crane's remarks.

We should be very glad to have some of these piers installed somewhere, so that the results of actual practice may be observed and recorded. We have had ample experience as to the value of elasticity in roof supports through the very wide use of the timber crib, the initial resistance of which is relatively small; it would seem highly desirable, therefore, to design and test a support which will combine the elasticity of the timber crib with enough rigidity to permit its carrying a pressure many times greater.

# Increasing Dividends Through Personnel Work

BY THOMAS T. READ,* E. M., PH. D., NEW YORK, N. Y.

(St. Louis Meeting, October, 1917)

PERSONNEL work is a term recently introduced to cover the great variety of activities in industrial work that deal with the human factor. Much attention has been focussed upon individual phases of personnel work, such as accident prevention, sanitation, "welfare" work, reducing labor turnover, and so on. In most cases these activities have been carried on as though they were distinct and separate fields; as a matter of fact, they are only different phases of the single problem of human engineering, and each has important and fundamental relations with the others. Thus, for example, high labor turnover is one of the most important causes of a high accident rate, and unsatisfactory housing conditions are an important cause of labor turnover. In the early stages of the study of these problems, it was necessary to take them up separately; the time has now arrived at which it is desirable to consider them as a whole in order to bring them into the right relation to each other. When thus considered as phases of an interrelated whole, they may be grouped as personnel work.

A discussion of the importance of personnel work as a means of increasing dividends in industry may be objected to by some as a too materialistic way of looking upon work that is humanitarian. Nothing could be more mistaken than such an opinion. Humanitarianism, *per se*, has no place in industry, and nothing is gained and much is lost in pretending that it has. The only reason why a corporation engages in business is to make profits, and any activity of the corporation that does not tend either directly or indirectly toward that end is poor business. It is equally poor business to overlook any activities that do tend indirectly to the making of profits. On this basis, personnel work can be justified as a corporate activity; there is no justification whatever for abstract humanitarianism from the business standpoint. One characteristic of the American workman is a general high level of intelligence, and it is an insult to intelligence to ask a man to believe that anything so impersonal as a corporation is really desirous of benefiting

* Technical Department, New Jersey Zinc Co.

him without advantage to itself. The inevitable result is suspicion and opposition. On the other hand, it is easy to make clear to the worker that he, as well as the corporation, benefits from personnel work, and thus win his support.

It must be kept in mind, as a major premise of this discussion, that through the rapid increase of invested capital in recent years and the much slower increase of labor supply, most corporations have already passed over from the position of buying labor to selling employment. The difference needs no emphasis, it requires only to be stated. The majority of important industrial organizations are now in the position where they need more capable men than they can secure, and it has become essential to hold out inducements for capable men to enter their employ, and to develop the capabilities of those they can get. Without going into details, it is my purpose to discuss briefly what has been and is being done and some results so far attained.

## Scientific Management

Since labor is the largest cost factor in nearly all kinds of production, it is obviously absurd to permit labor to be wasted in unproductive operations through faulty management. Much study has been given to this phase of the general question and a tremendous volume of literature on the subject has appeared; some of it controversial. It would require too much space to go into this subject here, and as it is impracticable to handle it both briefly and adequately, I will pass on after giving it first mention to indicate its importance.

## Accident Prevention

Accidents are a source of loss in industry for four reasons: (1) The injured man and his fellows who come to his assistance lose time from their work, and machinery is often shut down, thus interfering with production. (2) The employer, in nearly all States, is obliged to provide medical aid for the injured man and, if the injury causes more than a brief loss of working time, to pay wage compensation as well. (3) While the man is out some one must be hired to take his place, thus decreasing efficiency, increasing the labor turnover and, as will be shown later, increasing the accident hazard by the employment of a new man. (4) If there are many accidents in a given plant, employment there is less desirable, and the workmen are less efficient, since the fear of injury hampers their work. Accidents are wholly against the interest of employer and employee, since both lose by them and no one, in any circumstance, ever gains anything. Any progress that can be made in their prevention is a gain.

*Methods of Accident Prevention*

There are two classes of methods of accident prevention, mechanical and psychological. The former endeavors to remove accident hazards, the latter aims to teach the men to avoid them. The psychological may be subdivided into two classes which may perhaps be designated as amateur and professional, since the question of money reward is the distinguishing feature. In the former the emphasis is placed upon the use of posters, voluntary committees, public gatherings, moral exhortation, etc., and in the latter the chief reliance is placed upon some form of reward, either money or prizes, to the foreman of a gang, or preferably to the men themselves, for decrease in the accident rate. The first method is the one more commonly used, and it is important to discover, if possible, which is the better. In the accompanying diagram are plotted the records of seven companies that have kindly given me their figures. These are all "lost-time" accidents, minor injuries not being included. There is little uniformity as yet in methods of keeping records among the companies, but six of these companies record as a lost-time accident one in which the man does not return to work for the next subsequent shift. Company E records the accidents that caused more than 2 days' lost time, in accordance with State regulations. All the records are plotted on the basis of 1000 shifts worked, with the exception of company A, which keeps its records on the basis of 10,000 hr. worked. Most of the other companies work chiefly 8-hr. shifts but make no allowance for overtime, so that the discrepancy is probably not very important. The number of hours worked is, of course, the only exact basis upon which to make comparisons of accident records.

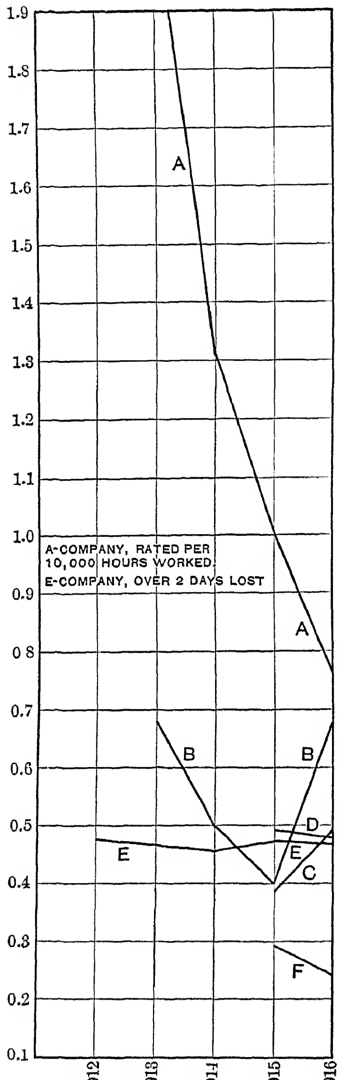



FIG. 1.—LOST-TIME ACCIDENTS OF SEVEN COMPANIES RATED PER 1000 MAN-SHIFTS.

Company B is a large chemical manufacturing company; C, D, E,

and F are smelting companies in widely separated parts of the country. All four use what I have designated as the amateur method of accident prevention, but F has recently supplemented this by the use of prizes. Company E may be taken as typical of this group. It began work with a qualified safety inspector in 1912 and now has a safety inspector, an assistant safety inspector, a clerical assistant, and an experienced man to train first-aid teams. Safety suggestion boxes and bulletin boards for safety bulletins are used, occasional lectures and moving pictures are given, weekly plant inspections are made, all accidents are investigated, the plant men are put through a safety class, and there are more or less active safety committees in the different departments of the works. Many thousands of dollars have been spent for safeguards to machinery and other dangerous places about the works, and yet in spite of all this no appreciable change has been made in the accident rate. This seemed so remarkable that I secured the permission of the company to make a further study of their recent accident records. This disclosed that of all their accidents (both minor and lost-time) 75 per cent. were due to the workman doing the work in an improper manner, taking unnecessary risks, or exhibiting simple carelessness. Of the remaining 25 per cent. nearly all, 22½ per cent., were classified as unpreventable; these were mostly minor accidents that a reasonable amount of care on the part of the workman would not have enabled him to foresee or prevent. In only 2½ per cent. of the cases was the accident due to any lack of safeguards, or dangerous conditions about the works. This is confirmed by Fig. 2, in which the total accidents in 1915 and 1916 are plotted by months in comparison with the number of men working on jobs that are new to them. The upper line shows the number of new men hired plus the men who have been transferred from one kind of work to another. It is evident that there is a close parallelism between the two curves. Besides showing that green men increase the accident rate, it indicates that the most important factor in accident prevention is the man himself, thus confirming the argument above. The sharp increases shown by companies B and C in 1916 are unquestionably due to increases in their working forces, thus rapidly increasing the number of green men. This will be further discussed under labor turnover.

Company A, on the other hand, is a mine at which the officials decided, on beginning safety work, that it was impracticable to employ safeguards and determined to rely almost wholly on the foremen and men themselves. A series of prizes was therefore offered, large enough to be worth working for, at first to the foremen and later to the men, to the gangs that showed the greatest accident reduction over monthly, half-yearly, and yearly periods. The results speak for themselves. This is a rather dangerous mine, being subject to sudden falls of rock from the roof and sides, which, as is well known, is the chief source of

accidents in mining. Before beginning accident prevention work, the accident rate at this mine was well above the average for all mines; it is now well below the average. It should be added that this company considerably increased its working force in 1916. It may be argued that the accident rate at this mine is so much higher than at the smelting plants that there is better opportunity for reduction, but this does not alter the essential fact that the conditions existing at each of these plants before safety work was begun were regarded as normal, two of them showed no important change in the accident rate, two showed a decided increase in

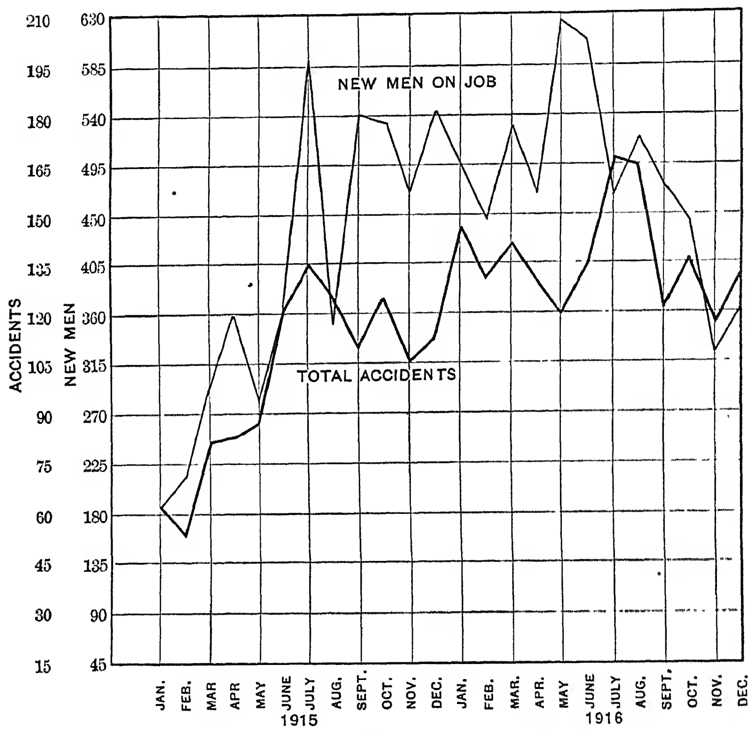



FIG. 2.

1916, and F, the only one that showed a decrease in 1916, adopted the use of prizes about that time.

Therefore evidence from the actual comparative records of companies, the analysis of the causes of accidents of one company, and the influence of the green man on the accident rate, all indicate that reliance for accident prevention must be upon the individual worker himself. This is also in accord with common sense. The workman is paid for speeding up his work and getting results; it is through these that he hopes to win promotion and increase in pay. If the only reward he gets for the avoiding of accidents is the preventing of injury to himself, he regards

safety bulletins as excellent tracts aimed at someone else; he is quite confident that his own skill and cleverness are such as to enable him to avoid injury. But the offering of substantial money rewards for the avoiding of accidents makes every man in the gang his brother's keeper. The rewards should not be too large, otherwise there is a tendency to conceal accidents, which is against the interest of the company, since infections due to slight injuries that have been neglected are likely to lead to a serious loss of time, require a good deal of medical attention and may require the payment of compensation. A small reward, handed over by the superintendent himself with a word of commendation, seems to produce the best results.

Before leaving this subject, I wish to record my personal opinion that the first method, which involves installing of mechanical safeguards, and which has behind it the pressure of State laws and the sales acumen of the manufacturers of safety devices, has in many plants been carried quite far enough or even overdone. Not a few guards are almost as dangerous as the conditions they are supposed to remedy, while the expenditure of considerable sums in, for example, the placing of toe-guards on platforms on which nothing is ever laid or beneath which no one ever walks can find little justification in business sense. I do not decry the use of suitable safeguards; they would be necessary even if for no other purpose than to demonstrate to the men that the company is sincere in its efforts to decrease the accident rate. But even those who insist most strongly on mechanical safeguards admit that they cannot be relied upon to prevent more than one-third of the accidents and the figures that I have given indicate that this is much too high an estimate. Therefore, to suppose that the problem of accident prevention can be solved through the use of mechanical safeguards alone is to proceed on a wrong hypothesis.

## Illness Prevention

So much attention has been focussed on accident prevention that the related problem of illness prevention has not received the attention it deserves. It has already been pointed out that the fundamental basis of personnel work is conservation of the labor supply. Fig. 3 shows the relative importance of accidents and illness as a source of lost time in industry, the figures being those of one company that has good medical supervision. It must be noted, however, that the figures for illness represent those men who reported that their absence from work was due to illness, and no method of confirming this was employed. As the lost time from illness is plotted on a 3-day absence or over basis, and that from accidents on a 2-day basis, it will probably involve no serious error to assume that the time lost from illness thus unrecorded counterbalances

the time lost recorded as due to illness when actually due to other reasons. It thus appears that the time lost from illness is, in the case of this company, four times as great as that due to accidents. On the average, the loss to the individual man is much greater, since the average time lost

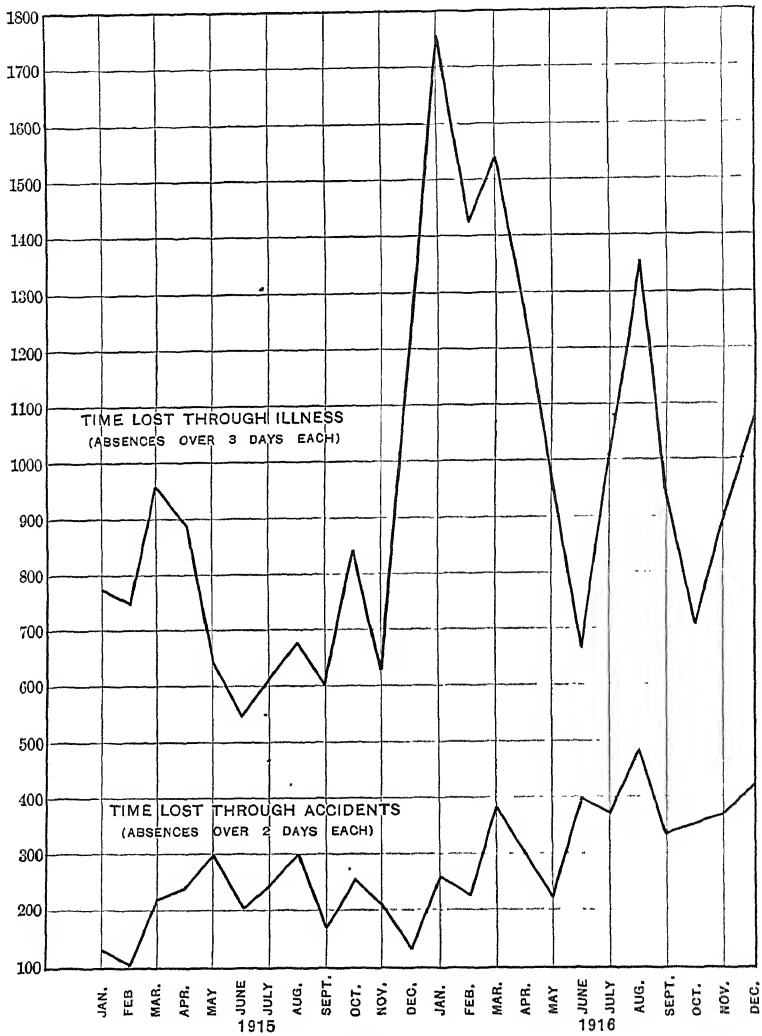



FIG. 3.

per accident was 2.53 days and the average lost per case of illness was 7.31 days. As in the case of accidents, when the man is absent another man must be supplied to take his place, and this both increases the labor turnover and the accident rate; in other words, it is a source of considerable loss to the company as well as to the man himself.

Almost every company nowadays has some provision for medical supervision of its personnel, and in many instances the equipment provided is quite elaborate. In practically every instance, however, this is employed only to take care of the man who is injured in the course of duty, and it is a common practice for the plant hospital to send men to their family physicians, after giving them any immediate attention they may require. The result, of course, is that in many cases they go home and apply home treatment, if any at all, and a case that might have been cured in a day or two results in a protracted absence from work. This state of affairs has its inception in the idea that it is inadvisable for the plant physician to do anything that might decrease the practice of local physicians. It is clear that a corporation is justified in doing anything that is legal and tends to the improvement of its business, and it is equally clear that the preventing of illness among its workers does tend toward the improvement of its business. I look forward confidently to the day when the medical departments of large corporations will not only render medical attention to injured men but will give them any other medical attention they may require. The medical officer should also be required to make frequent inspections of all working places in order to detect and remedy any conditions that may be inimical to the health conditions of the men. In short, the men should receive as careful attention as is given to the machinery, to insure that they are in good condition and operating efficiently. Roughly speaking, it costs as much to operate a man as it does a 40-hp. motor, and careful study to insure proper adjustment and proper operating conditions is at least as worth while with the man as with the motor. Proper plant sanitation is therefore essential; this will be referred to from another aspect in discussing "welfare" work.

## Labor Turnover

It is only of recent years that the importance of this factor has been realized, and we are still feeling our way as to its proper treatment. Large corporations developed slowly from the primitive stage of industry in which each employer was his own foreman, and there grew up a tradition that in order to preserve discipline the foreman must be able to employ and discharge his own men, in face of the fact that discipline is nowhere more rigidly enforced than in the army where the sergeant has never had the authority to enlist or discharge a private. With the coming of compensation laws it became necessary to make a preliminary medical inspection of the new employee, which made a central employment office desirable. Every plant of any size has a central storehouse where the foreman sends for any supplies he needs and to which he returns any he cannot use, and it is now becoming clear to employers that labor merits as much care and economy in its handling as supplies.

Many plants have taken away from the foreman the privilege of hiring and discharging his employees without interfering in the least with discipline, but with the result of saving a good deal of the foreman's time. I recently heard of one case in which the foreman complained that men who came to him to get his permission before requesting the employment office to transfer them to another department were wasting his time. The system of having all hiring and discharging of men done by an employment manager has now been adopted by a number of corporations, with most satisfactory results. One concern, of only modest size, reports that through the use of an employment department the company, in the first year, decreased its labor turnover 48 per cent., decreased the hours worked 9 per cent. along with a 10 per cent. increase in output, introduced a bonus system for regular work that cut down its lost time, remedied unsatisfactory conditions that had previously been undetected in a number of departments, and has done away with strikes and wage disputes.

So little attention has heretofore been given to the cost of hiring men that it may be well to indicate what the sources of this cost are. (1) The cost of hiring the man, interviewing him, making a medical examination, placing him on the pay roll, sending him to his foreman, and getting him assigned to work. (2) The difference between the standard amount of work and that which he does while getting used to his job. (3) The extra materials which a new man spoils. (4) The extra supervision required by a green man. (5) Interference with the work of fellow-workmen. (6) Accidents caused by the green man. Various executives have computed the total amount of these costs for their own conditions and the results range from $25 to $100 as the cost to a company of bringing a green man up to the stage of a productive workman. To this must be added the cost of those men who work only for a short time and then leave. During 1916, most companies have been increasing their working forces and the results of those who keep careful records show that the ratio between the number of men hired during the year and the actual increase in the working force ranges from 4 to 1 to 15 to 1. In most cases, the number of men who must be hired in order to keep a working force at normal strength varies from 100 to 200 per cent.; in other words, if a company maintains an average working force of 1000 men it would expect to hire (and lose) from 1000 to 2000 men in the course of the year, under present conditions. It is evident, therefore, that the costs involved are enormous, and no phase of personnel work will better repay study than this.

### *Methods of Reducing Labor Turnover*

The most important and effective method is the creation of an employment department under a capable manager and not, as is so often

the case in starting personnel work, turning it over to some one in the organization who is at the time without much to do to "see what he can do with it." An electrician who was not busy might be able to take charge of a carpenter shop, for his general knowledge would help him, but a man who has never given any time or thought to personnel problems is not likely to be able to accomplish anything worth while without delaying a long while to gain the necessary knowledge and experience, even if he is temperamentally fitted for the job. Such a man should rank as an assistant superintendent if he is to perform his work effectively. As has already been indicated, the fields of accident prevention and labor turnover are so closely connected that it is desirable that both should be under the supervision of one man.

The other methods of reducing the labor turnover may best be discussed under welfare work, since the primary purpose of such work is to bring about conditions that will induce men to remain in the employ of the company, and thus cut down labor turnover.

## Welfare Work

It has just been stated that the primary purpose of welfare work is to decrease labor turnover. It may be put in another way, as an effort of the company to provide for the men as a whole comforts and conveniences that they could not individually provide for themselves except at a prohibitive cost. The other alternative is the course pursued by Mr. Ford of paying an abnormally high rate of wage. The latter is much the simpler and easier way, but it is hard to justify it from the standpoint of political economy, and Mr. Ford has found it desirable to utilize both methods. It is obviously better, from the standpoint of society, for a company that is selling employment to offer a man in return for his labor a combination wage made up of the most desirable proportion of cash, which he expends as he desires, and "kind," which in this case consists of those things which the company can provide to better advantage and at less expense. A good example of this is the United States Army, which pays both officers and men in cash, provides living quarters or else makes cash allowance for them, and either provides subsistence or sells supplies at a lower cost than they can be bought in the open market. The necessity for such work is not always evident in industry, for manufacturing plants are, or were in former years, commonly established in the neighborhood of large cities, where the employee is expected, and usually prefers, to fend for himself. Following this tradition, operations in remote places, such as mining companies, ordinarily provided nothing for their employees beyond the bare necessities of existence, and these too often of inferior quality and at an excessive price. The company store, for example, in itself a desirable institution, was so grossly abused that it has become discredited. Recently it has

become the fashion in starting large new enterprises to build industrial towns *de novo*. This insures freedom from the complexities of local politics and permits the organization to choose the conditions under which it will operate, instead of attempting to adjust itself to conditions already existing, as well as to secure the necessary real estate at a reasonable cost. Houses, water supply, roads and pavements, sewers, electric light, telephones, kindergartens, schools, libraries, clubhouses, banks, etc., are needed at once; to wait for private initiative to provide them would involve expensive delay, while to depend on municipal endeavor resembles the Irishman who said of his new boots that he could get them on after he had worn them awhile. It is so obviously more economical for the company to provide these things at once when they are needed that it is becoming general practice. Unfortunately, the management of some companies has not been intelligent enough to perceive that to continue control over such public utilities beyond the stage in which it is essential is an unwarranted interference with the functions of public government that is certain to produce trouble in time. This is a perfectly natural state of affairs, just as parents, having raised their children from babies, are inclined to continue an undesirable degree of control over them when they have reached years of discretion. In both cases the tendency should be recognized and an effort made to correct it.

Of recent years, it has been recognized that the provision of such forms of labor compensation is not only a necessity but is also good business. The statistics of those companies that keep adequate records show that the chief element in labor turnover is furnished by the men without families, who board. In an industrial town where good homes are available at a reasonable rent, good schools, well kept streets, adequate stores that sell supplies at reasonable prices, and where the general tone of the community is good, steady workmen with families will be glad to make their permanent homes. The company is therefore able to buy productive labor at less cost to itself and can afford to invest capital to bring about so desirable a condition, just as it invests capital to lower its other operating costs. It would obviously be impossible to attempt any complete discussion of this topic here, but I should like to touch on two or three points that seem to me most significant.

### *Housing Conditions*

The cost of maintaining a roof over his family's head is the largest single item in the expenditures of a man of moderate means, and therefore calls for a corresponding amount of study. The policy pursued by some industrial organizations seems to me unlikely to produce the most desirable results. In building a new town, a standard type of house is designed, perhaps by an architect who has never lived in anything but

a hotel or an apartment house, and is correspondingly unfamiliar with the needs and desires of a working man's family, and houses of this type are built in long rows, like peas in a pod. To the cost of house construction, the company adds the cost of land, streets, sewers, etc. and dividing this by the number of houses obtains the average cost of a house, on which the rent is based. If the company sells lots, a similar procedure is followed, with the result that real estate values are inflated (for the initial cost of streets and sewers is disproportionately high) and rents become high. A better method would be to carry all public utilities furnished by the company in a separate account, with the understanding that as early as possible they shall be taken over by the local government at cost price and that in the meantime interest upon the sums thus invested shall be credited to the company as a rebate on local taxation. The former method makes the company defeat its own ends, which are to provide good homes for its employees at a minimum price. It is evident that it is most desirable for the employee to own his own house. This is not now difficult for a man who is regularly employed, since there are reputable firms that will not only sell building material on credit to the owner of a lot, but will actually loan him enough money to pay the major part of the cost of construction. On this basis it would be profitable for a company that owns sufficient land to give to a steady employee a deed to a lot, with the proviso that he must build upon it a house to cost over a fixed minimum sum within a reasonable length of time. No additional capital investment is thus required of the company, and its return on the original capital invested in the land comes back to it in a decrease in the tax rate through the increase in the taxable valuation of the town. If all the houses in an industrial town have to be built by the company, the amount of capital thus tied up is enormous, returns on the capital are slow, and there is a possibility of an enormous shrinkage of values if for any reason the venture is not a success. Many companies have held on to real estate for the purpose of controlling saloons, gambling houses, and the like, but this could be easily taken care of through deed restrictions. Not the least advantage of such a plan is that it would do away with the undesirable uniformity of industrial towns, where a man can secure any number of houses renting at $8 or $12 a month, all alike, but cannot find any houses renting at $20 or $30 a month. The amount of money tied up in houses of the latter type is so much greater that supplying them in any large number is undesirable and it is also more undesirable, from the tenant's standpoint, to have them all alike. It is so obviously to the company's advantage to have its foremen and subforemen well housed that this phase of the problem is worthy of the most careful study, and if the man who has charge of personnel work is not in charge of housing he should at least be constantly consulted regarding it.

### *Change Houses*

Charity begins at home, but efforts to improve the morale of the workmen should begin in the plant. Unless the workmen are provided with clean, commodious, and well-designed change houses, ample toilet and bathing facilities, in connection with their work, it seems idle to attempt through welfare workers to raise the standard of living in the homes. Toilet facilities should be of the modern type that do not lend themselves to the transmission of disease. It is above all important that such conveniences shall not only be well built but also well kept. The care of a change house is too frequently considered a suitable post for a partly disabled or superannuated man, with the result that the care given is little more than nominal. Keeping such facilities in good condition requires plenty of hot water, scouring soap, and elbow grease, and the man in charge should be vigorous enough to apply all of these effectively.

### *Conveniences for the Workmen*

The theory used to be widely held that children and all other subject classes in the community should be made to do all sorts of things they did not want to do in order that they might learn discipline, but it is now pretty generally recognized that people should be allowed to do what they want to do if there is no valid reason against it. Nevertheless not a few foremen regard acceding to requests from the workmen as "coddling them." I have already made the point that companies are now selling employment instead of buying labor, and an attitude of mind that is perfectly practicable under the latter condition is not practicable under the former. A large corporation in Colorado has created a complex organization of which the purpose is to afford opportunity for consideration of the wishes of the workmen by the management, and this seems a step in the right line of progress. If the workman believes that those from whom he takes orders are reasonable human beings who are willing to listen to any request and either grant it or explain why it is not practicable to do so, a tremendous advance has been made toward awakening that spirit of coöperation which must be the corner stone of industrial progress. In every company there are many little ways in which the comfort and convenience of the workman can be served, and the money thus expended will return big dividends. In his interesting autobiography, Charles Francis Adams remarks that the Adams family were always able to do a gracious thing in an ungracious manner; and in this respect many industrial corporations resemble the Adams family.

*Necessities of Life*

It is unfortunate that the "company store" has been brought into such discredit, for it is an institution that had many possibilities for good. The proverb says that the best thing to do with a dog that has a bad name is to hang him, and a similar course of action seems to be indicated for the company store. In some places, employees have formed coöperative buying associations that serve to accomplish much the same results. The coöperative gardening schemes now in full swing throughout the country are an excellent method of enabling the employee to keep down the cost of food, and a man who is busy in his garden has no time to loiter around saloons and other gathering-places of discontent. The psychological effect of such work is not the least part of its value. The man who is willing to devote some of his spare time to labor that is not immediately profitable but is expected to yield returns in the future is likely to exhibit the same spirit in the work that he does for his employer.

*Schools, Social and Religious Organizations, Politics*

These are all forms of activity into which employers may venture only with extreme caution. The schools of a community may be below grade and the company may wish to improve them. This can best be done through the instrumentality of sagacious employees who act in their private capacity as citizens. Social work, where it is done, is commonly in the hands of trained welfare workers. The kindergarten is an excellent means of introduction to the homes of the parents, and the Boy Scouts and Camp Fire Girls organizations are similarly useful among the older children. The great problem in all such activities is to leave the apparent initiative with the public. We all remember the lady in Stockton's "Associate Hermits" who wanted everybody to develop his individuality, and what a nuisance she made of herself. The general public can be "uplifted," but it must be approached with tact and, above all, it must be allowed to believe that the uplift comes from within.

Religious affairs are particularly difficult because religion is a matter of sentiment rather than reason and is therefore not open to logical analysis. The best thing to do is to leave religious matters alone, but unfortunately they will not leave industry alone. Religious festivals frequently interrupt the work of a plant for days at a time, quarrels in a church organization frequently interfere seriously with the attention of workmen to their work, and differences in religion often lead to friction between the members of a gang of workmen. It is obvious that a difficult situation should not be further complicated by permitting a person of a strongly religious turn of mind to engage in personnel work.

In politics, there lurk so many pitfalls for unwary feet that I will not discuss it further than to say that industry would be glad to leave politics alone if politics would leave industry alone.

### *Educational Work*

The varieties of educational work being done in different places are almost too numerous to mention. I have not had opportunity to make any thorough study of it, but have gained the impression from what I have seen that in many places the results are somewhat disappointing when compared with the amount of attention given to it, and that this is due, in some cases at least, to the use of volunteer teachers who are not sufficiently skilled in the art of teaching. Teaching of English to foreigners is popular just now and there is a good deal of weight behind this desirable movement. Some concerns have well organized schools for training the workman before letting him loose on the plant; one concern puts an experienced man in charge of a gang of new men and rates his pay upon the output of the gang. The apprentice system is good wherever conditions are suitable for its use. It is obvious that any education of the workman improves his efficiency and benefits his employer unless the education is so badly directed as to make him discontented with his work and to want to get into a "white collar" job. When, as now, a bricklayer can earn more money than a college graduate, it is curious that this desire for "white collar" jobs should exist, but it does and it needs to be reckoned with in directing educational work.

### Conclusion

Labor has become the largest cost factor in productive industry and seems likely to continue its relative increase. The ideal of industry is a laborer who is sober, efficient, and steady. The first is in progress of attainment by the advance of prohibition of the liquor traffic. Efficiency in the workman is attained through better management and training of the worker. Lost time is overcome by the prevention of accidents and illness, and through a great variety of means that bear indirectly, as indicated above, on the labor supply. Industrial accident prevention looms large in the public eye through the recent enactment of compensation laws, but as a source of lost time to the plant and lost wages to the worker it is much less important than illness. Health insurance is now being urged in many quarters, and if it comes about we may expect a transfer of emphasis from accident to illness prevention. Lost time through religious holidays and other personal activities, or inactivities, of the worker is much more important than either; their relative values being something like 1, 4, and 10, in most cases. The only practicable

means of dealing with the latter seems to be the offering of a bonus, over and above wages, to the workmen who lose no time each month. The underlying purpose of personnel work is to conserve the labor supply and decrease the net cost of productive labor. Personnel work is human engineering, it requires natural aptitude and special study and is not a field for the well-meaning amateur, especially the amateur who is unfamiliar with the peculiar problems of the industry in question.

## Discussion

C. W. Goodale, Butte, Mont.—While there are many points in Mr. Read's paper which I am not prepared to discuss thoroughly, I am inclined to believe that difficulties are involved in the establishment of an employment bureau which shall have exclusive control over the employment and discharge of workmen.

I suppose you have heard a great deal about the so-called "rustling card" at the works of the Anaconda company. That is a very innocent sort of card, although the I. W. W. organization has made a great deal of noise about it. An applicant for a job is handed a blank on which he tells his nationality, his age, whether married or single, and the place of his last employment. The company then writes to the place where he says he was last employed to find out whether he is telling the truth; we do not ask the previous employer whether the man is blacklisted or has anything against his record, but simply, "Was this man formerly in your employ?" A great hullabaloo has been made about this practice by some of the men in Montana who are afraid of their own records; but out of 53,000 applications for employment received by the Anaconda company since the employment bureau was formed, only 647 were rejected, which does not indicate any great hardship on men who are willing to have their records examined.

When a man is given a rustling card he is privileged to go to all the mines of the company and ask for work; then it is up to the foreman to inquire about his line of work, and find out whether he is a well qualified miner. Thus the actual employment is settled by the foreman. As for discharging workmen, it would be fatal to discipline if the authority to discharge an incompetent or careless man is taken away from the foreman or shift boss and transferred to some bureau.

In connection with safety work, we have made a great effort to reduce our accidents, and have thought it necessary to inflict a penalty when a man is guilty of a flagrant violation of safety rules. In such cases the shift boss was authorized to lay the man off for 7 days, and for a second offense for 14 days. This has been made a great ground of complaint, because, as they say, it gives a foreman or shift boss a chance to lay off a man just out of sheer spite, without relation to any safety rules. The same spiteful disposition could have been exercised before we ever

established that rule, but in our resumption of work lately, the company decided to suspend the infliction penalties, and our safety engineers are in despair about it. They do not know how they will be able to make the men look out for themselves and obey the safety rules of the company, if they are not going to suffer any penalties for infractions of those rules. While it may be that the fear of punishment is not a stronger motive than the hope of reward, it has been rather difficult to arrange any bonus system to apply to the rank and file of the men. It might be possible to arrange a bonus for the foremen if they keep their accident records down, but as for the men themselves, there are 10,000 employed in the mines of the Anaconda Copper Mining Co., and it would be almost impossible to invent a bonus system applicable to each individual man. The Anaconda company has made use of all the methods of accident prevention which Mr. Read has described in his paper, and has succeeded in effecting a considerable reduction in the accident rate.

The hazards in its metal-mining department are shown by the following figures for the year ending June 30, 1917: There was one fatal accident in 114,942 shifts worked, and one serious, non-fatal accident in 12,771 shifts—serious accidents being those which involve a loss of more than 14 days. I believe the above records will compare favorably with the figures in other lines of industry.

H. M. Wilson, Pittsburgh, Pa.—On looking hastily over Mr. Read's paper, I saw some important points which seem to have been overlooked by the last speaker, and another feature which seems to have been incorrectly accentuated. I do not understand that Mr. Read proposes the appointment of an individual who shall be independent of the mine management, for the duty of discharging men. He suggests an employment agency; I do not believe he had any idea of a discharging agency. None of us will question for a moment that the latter power in the hands of anyone but the foreman, superintendents, and managers could be other than a serious menace to the business.

Another feature of the paper which has impressed me is the author's endeavor, based on rather limited statistical data, to determine the relative effect of what we may call propaganda, being the educational effort to induce officials and mine workers to take an interest in their own safety, as compared with money encouragement. Mr. Read's data, limited as they are, would seem to indicate that bonuses paid to mine officials and men for good accident records had a better effect toward keeping down the accident rate than almost any amount of talking. With that I heartily agree. Personally, I have serious doubts as to the efficacy of any form of punishment. Encouragement, in some form which they appreciate, is what men need. Bonus systems have worked splendidly in many mines with which I am acquainted. There is a strong ri-

valry in some mines, between various foremen, for the showing of a good accident record, not merely for the money bonus they will get but for the satisfaction of crowing over their rival fellow officials.

In the line of business in which I am engaged—casualty insurance—we have found by positive results that a money bonus is the strongest possible incentive to the mine owner, the manager and the officials beneath him. The system of merit rating, whereby the premium on accident insurance under workmen's compensation may be reduced for each element of safety precaution introduced into the mine, has been a wonderful incentive for improving mining conditions. When the mine owner gets back in the form of a premium rebate a sum of money which would almost pay for the improvement he has made, he is more likely to pass it along in the form of a bonus.

Another feature of Mr. Read's paper which interested me was the relationship between health and accidents. The two are very distinct. Bad health may result from accidents; accidents do come from ill health. Either one of them affects the amount of labor your employees can do for you; therefore it is well, while considering the prevention of accidents to give serious attention to the preservation of health, for surely a healthy workman with a strong body and a clear mind can protect himself better than an ill one.

C. M. Haight, Franklin, N. J.—The New Jersey Zinc Co. has an employment agent at Franklin from whom we requisition men for the mine. The foremen have the right to reject any man sent in by the employment agent, if they so desire, and after he has been employed they have the right to discharge him; they do not discharge him from the organization, however, but merely from the mine department. The employment agent then has the option of turning him over to some other department, to see if he will fit in there, and if not, to discharge him from the organization. Mr. Read told me that he intended to bring out the fact that he proposed to allow a foreman to discharge a man from his department only, not from the organization.

J. A. Ede, La Salle, Ill.—A member of this Institute, Mr. T. F. Noon, President of the Illinois Zinc Co., who, owing to illness, is not present today, has worked out some of the suggestions presented in Mr. Read's practical and interesting paper. Bonuses have been paid to the coal miners employed by the Illinois Zinc Co. regardless of the fact that they are members of a highly organized union. They have been privileged, together with the factory hands, to participate in the distribution of a bonus, the increment of profits from the metallurgical department; that there has been no strike in either of these mines, although they have been operating for 17 years, speaks in favor of the policy pursued.

The formation of an employment department is a subject presenting many angles, and its introduction into the executive department should be determined by individual environment and conditions, whether for the employment of men in factories or mines. In coal mining the manager's powers are already circumscribed within very narrow limits. The fact that he has the privilege of engaging his men forms one of his strongest factors. If he is deprived of this, I am afraid his office would lose much of its moral influence and power.

I fully appreciate that there are places where an employment bureau or committee would be very serviceable. Before a man is employed to perform certain duties, he should be examined to know whether he is physically as well as intellectually competent to perform the duties required of him.

W. Y. Westervelt, New York, N. Y.—Mr. Read's remarks on the relation of the draft to the present labor situation remind me that when this topic was under discussion by the War Minerals Committee we decided that, although the committee might take no formal action, its individual members personally ought to urge employers of technical skilled workmen to consider very carefully whether such employees were likely to be more serviceable in the fighting ranks than in their present occupations. The first need, of course, is that a powerful army shall be created, and if that necessitates the drafting of the skilled workmen of the country, this must be done. It is to this end that the draft officials have bent all their energies. The employer of skilled workmen, on the other hand, is better able than the draft officials to decide whether or not a particular group of specially qualified men is likely to be more useful at home than in the army. I do not refer so much to individuals, because no one man is positively essential to any enterprise; but if an employer feels certain that his output of materials needed for the war is likely to be curtailed by the drafting of a considerable number of his workmen, I believe it is his patriotic duty to urge their exemption.

R. M. Catlin, Franklin, N. J.—Like many other mine managers, I suppose, it has come to my knowledge that some of our men have been called upon to serve in the army. When I sought to exempt them, I found it was necessary to file an affidavit to the effect that they could not be replaced without serious material loss to the State. How many such men exist in this world? From the President to the office boy, every man can be replaced, if you know where to get him. On the other hand, it is also clearthat every man whom I should cause to be released from military duty has to be replaced in the army by someone else. It would have been better if the wording of the regulation had been clearer. I

have no doubt what it means, but to require an affidavit that any man is absolutely essential to any work, is a pretty tall order. For that reason we have refrained from making many of those applications, although I knew it would be very difficult to replace certain men who have gone.

W. Y. Westervelt.—I appreciate the difficulty described by Mr. Catlin in deciding whether the service of any particular workman can or cannot be replaced without serious material loss to the State. The wording of the exemption clause of the draft law is unfortunately obscure, but I take it as applying to all the men of a particular group; thus, if all expert chemists now employed in metallurgical works were thrown into the ranks, our smelters might come to a standstill, or if all our engineers were sent to Europe, we could not keep our own industries going. This idea is entirely in line with the President's message in which he says that the miner stands where the farmer does, and that if either slacks or fails, the efforts of soldiers and statesmen will be in vain. I have felt it my duty to put my name to this affidavit which has been found so troublesome, and I expect to have to do it again.

T. T. Read.—May I have a moment to reply to the various points that have been raised? We would all agree that outside interference in the management of industrial enterprises is wholly undesirable. What I have urged is a better organization within the enterprise itself. The employment manager is the direct representative of the employer and wholly under his direction.

Replying to Mr. Goodale, I would say that the objection he has raised is the first one made in every instance when the placing of the hiring of men in the hands of an employment manager is proposed, and the best possible answer to this objection is that in no instance has practical experience shown it to be justified. I possibly failed to make it clear that when hiring is done by an employment manager the foreman has just as much authority as he had before; he does not need to accept a man sent him by the employment office unless he wishes to, and he has complete authority to fire the man out of his own gang. But he is relieved of the labor of going out and getting men when he needs them; they are sent to him by the employment office, which has investigated them and concluded that they are apparently competent to do the work he wants done. The foreman has neither time nor facilities for doing this; usually he looks a man over, takes his word for what he can do, then frequently finds that the man cannot do it, and fires him. In this way it is easier for an incompetent man who is a good talker to get a job than it is for a competent man who has not the "gift of gab."

Under the plan I am advocating, if a foreman finds a man unsatisfactory he discharges him out of his gang, by sending him back to the employment agent. But a foreman should not be allowed to discharge

a man from the company any more than he should be allowed to throw away material that he cannot use. A man discharged by a foreman often goes away with the feeling that he has not been given a square deal; meanwhile some other gang might have been able to use him to good advantage, and the company has thus acquired an enemy instead of a loyal and efficient employee.

I wish to reiterate that while many companies have been slow to recognize the need of an employment manager, this system has been adopted with success by a great number of industrial organizations. Three that will be familiar to you are the American Rolling Mills Co., the New Jersey Zinc Co., and the Jeffrey Manufacturing Co. One of the great advantages of having an employment manager is that the men feel that he is the direct representative of the employer. In the early days of industry, the employer hired his men and knew them as Bill, and Jim and Jack, but nowadays the heads of some big industrial organizations have never even visited some of the smaller properties they control. The workman deals with the subordinate of a subordinate of the subordinate to the *n*th degree, and it is no wonder that he feels that he has no means of bringing his viewpoint to the attention of his employer.

It is natural that men in charge of operating departments should say that any new arrangement like this will not work. The operating man is characteristically conservative and when the research department of his company proposes a new method or new equipment he invariably replies that it will not work. But if the process is sound and it has been properly studied, it will work; and the same holds true of the scientific study of employment problems. These problems are governed by natural laws, and what we need is more scientific study and exact information, instead of the preconceived opinion, bias and prejudice that is usually applied to them. Labor problems are not new; in 1349 the Parliament of Great Britain enacted arbitrary laws to deal with the labor shortage that then existed, and we have been trying to handle labor problems in an arbitrary way ever since. It is not complimentary to industry that it has not learned more in six hundred years of experience. The relations of capital and labor are governed by natural laws and as soon as we acquire the necessary amount of knowledge and skill we shall be able to solve these problems as they arise, and even to prevent them from arising.

# Influence of Base Metals in Gold Bullion Assaying*

BY FREDERIC P. DEWEY,† PH. B., WASHINGTON, D. C.

HAVING shown[1] the difficulty of assaying so-called cyanide bullion and the extreme variations often found in the results, an investigation was undertaken to discover, if possible, the causes of these variations by a series of tests upon the specific action of various metals during cupellation.

*Zinc.*—In order to test the effect of zinc, 48 assays were made in sets of 6, 2 rows of 3. The charge for the middle cupels consisted of 500+ mg. Au, 1,125 mg. Ag, 10 mg. Cu. To the end cupels 50, 75, 100 and 125 mg. Zn were added, and a set of each proportion was cupelled with 4 and 8 grams Pb. The cupels were assayed and showed the following gold contents in milligrams, the figures being arranged as the cupels stood in the muffle:

TABLE 1

| | 50 Mg Zn | | | 75 Mg. Zn | | | 100 Mg Zn | | | 125 Mg Zn | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 4 grams, Pb ........ | 0.28 | 0.28 | 0.28 | 0.29 | 0.27 | 0.34 | 0.33 | 0.28 | 0.41 | 0.31 | 0.30 | 0 36 |
| | 0.36 | 0.33 | 0.34 | 0.31 | 0.31 | 0.29 | 0.35 | 0.37 | 0.42 | 0.41 | 0.34 | 0.38 |
| 8 grams, Pb ......... | 0.46 | 0.43 | 0.42 | 0 38 | 0.36 | 0.43 | 0.35 | 0.34 | 0.41 | 0.33 | 0.37 | 0.37 |
| | 0.54 | 0.47 | 0.59 | 0.44 | 0.43 | 0.42 | 0.38 | 0.39 | 0.41 | 0.57 | 0.38 | 0.46 |

In many instances the zinc does not appear to affect the cupel gold, but various exceptions are apparent, generally in the cupels in the back row. With 100 and 125 mg. Zn, in some cases the Zn burned with its characteristic flame before it could alloy with the other metals. Also, less fusible oxidized lumps were observed during the cupellation and some cupels showed a scum.

In order to lessen the premature burning out of the Zn, an alloy of 2 parts Ag to 1 part Zn was used. A uniform charge of 500+ mg. Au,

* Published by permission of the Director of the Mint.

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[1] Assay and Valuation of Gold Bullion, *Trans.* (1909), **40**, 780–797.
Annual Report of the Director of the Mint, 1909, 25–36.
Sampling of Gold Bullion, *Trans.* (1912), **44**, 853–879.
Treasury Department Document 2672.
Cadmium and Nickel in Cyanide Bullion, *Engineering & Mining Journal* (Apr. 13, 1912), **93**, 733.

300 mg. alloy (containing 100 mg. Zn), 925 mg. Ag and 20 mg. Cu was used in sets of 9, 3 rows of 3. One set each was run with 4, 5, 8 and 10 grams Pb. In the table, the first column gives the relation of the cornet weight to the amount of Au taken and the second the amount of Au found by assaying the cupel, including beads, as well as Au actually absorbed by the cupel.

TABLE 2

| 4 Grams Pb | | 5 Grams Pb | | 8 Grams Pb | | 10 Grams Pb | |
|---|---|---|---|---|---|---|---|
| Cornet, Mg | Cupel, Mg | Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. | Cornet, Mg | Cupel, Mg |
| 0 12+ | 0.12 | 0.35+ | 0.18 | 0.58− | 1.09 | 0 08+ | 0.31 |
| 0.29+ | 0.20 | 0.41+ | 0.19 | 0.14+ | 0.36 | 0.11+ | 0.28 |
| 0.15− | 0.48 | 0.16+ | .... | 0.25+ | 0.28 | 0.11+ | 0.34 |
| 0 22+ | 0.35 | 0 22+ | 0.37 | 0.18+ | 0 38 | 0.34− | 0.82 |
| 0 32+ | 0 24 | 0.34+ | 0 27 | 0.08+ | 0 47 | 0.02+ | 0.48 |
| 0 33+ | 0 23 | 0.16− | 0 54 | 0.19+ | 0 35 | 0 06− | 0.50 |
| 0.02− | 0.34 | 0.25+ | 0.27 | 0.03− | 0 45 | 0 04+ | 0.51 |
| 0 25+ | 0.26 | 0 30+ | 0 32 | 0.04− | 0.54 | 0 14− | 0 60 |
| 0 23+ | 0 24 | 1.79− | 2.43 | 0 09+ | 0 43 | 0.11− | 0 50 |

In a second set, 500 mg. of the alloy, containing 166 mg. Zn, and 790 Ag, were used, giving the following:

| 4 Grams Pb | | 5 Grams Pb | | 8 Grams Pb | | 10 Grams Pb | |
|---|---|---|---|---|---|---|---|
| Cornet, Mg | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. | Cornet, Mg | Cupel, Mg. | Cornet, Mg | Cupel, Mg. |
| 4 80− | 5.01 | 1.97− | 2 10 | 0.06+ | 0.37 | 0.07+ | 0.30 |
| 0.31+ | 0.14 | 0.19+ | 0.24 | 0.25+ | 0.38 | 0.04+ | 0.34 |
| 1 80− | 2.17 | 0.01− | 0 42 | 0.23− | 0.75 | 0.21− | 0.69 |
| 0 21− | 0.62 | 0.42− | .... | 0.17+ | 0.39 | 0.24− | 0.64 |
| 0.25+ | 0 25 | 0.19+ | 0.28 | 0.17+ | 0.42 | 0.13− | 0.44 |
| 0.39− | 1.02 | 4.08− | 4.46 | 0.28− | 0.41 | 2.04− | 2.44 |
| 1.32− | 1.70 | 0 05− | 0.34 | 0.51+ | 0.46 | 0.34− | 0.79 |
| 0 19+ | 0.22 | 0 10+ | 0.35 | 0.13− | 0.51 | 0.18− | 0.49 |
| 4.41− | 4 63 | ...... | 5.71 | 0.10+ | 0.48 | 0.44− | 0.93 |

In these tests the pyrometer registered about 950° C. The figures are arranged as the cupels stood in the furnace.

From these figures, it is seen that Zn may be a most disturbing element. In various cases more or less infusible oxidized lumps were observed floating on the bead and these lumps entangled metal. In all probability this was due to the extreme preferential solubility of Au and Ag in Zn, the immiscibility of the resulting alloy with Pb, and its lower specific gravity; from all of which the Zn-Au-Ag alloy was formed, came

to the surface of the bead and the Zn was oxidized without sufficient Pb to liquefy the ZnO. Eventually some of these lumps inclosing Au and Ag reached the cupel.

In a final comparison, the Ag-Zn alloy was used to introduce small amounts of Zn, three proportions—62.5, 85 and 125 mg. of alloy—being used. Each proportion was cupelled with 4, 5, 8 and 10 grams Pb, in sets of 9 cupels, 3 rows of 3, the middle cupel charge being without Zn as in the first test. This gives 36 rows, but 2 rows were lost. The 68 cupels with Zn yielded 21.39 mg. Au, and the 34 without, 10.70 or substantially half as much, thus showing that a small amount of alloyed Zn has no effect upon the cupel gold. However, the ZnO was not always properly absorbed by the cupel. With 125 mg. alloy there was a distinct tendency for the Zn-Au-Ag alloy to come to the surface. Curiously, this was also shown with the small amounts of alloy when 10 grams Pb were used. Evidently, the longer time required to oxidize such amounts of Pb offered sufficient opportunity for the Zn-Au-Ag alloy to form and gather at the surface. Therefore, the use of larger amounts of Pb to slag off Zn is not advisable.

*Cadmium.*—Having shown[2] that Cd is often present in cyanide bullion, a few sets of 2 rows of 3 cupels were run, replacing the Ag-Zn alloy by Ag-Cd. One-half of the results obtained with each amount of Pb and Cd are given. The other results were similar.

TABLE 3

| 4 Grams Pb, 166 Mg Cd | | 10 Grams Pb, 100 Mg. Cd | |
|---|---|---|---|
| Cornet, Mg | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. |
| 0.60+ | 0.19 | 0.10+ | 0.38 |
| 0.57+ | 0.17 | 0.09+ | 0.38 |
| 0 60+ | 0.16 | 0.14+ | 0.38 |
| 0 58+ | 0.15 | 0.08+ | 0.44 |
| 0.64+ | 0.18 | 0.03+ | 0.48 |
| 0.58+ | 0.19 | 0.04+ | 0.44 |

Aside from four beaded cupels, the results were uniform and very satisfactory, showing that Cd cannot be the cause of the wide variation so often found in cyanide bullion assays. No infusible lumps were observed in the cupelling beads, but in the 166 mg. Cd, 4 Pb sets, the cupels were heavily coated with the characteristic dark brown oxide of Cd.

The low Zn alloy runs were duplicated with the Cd alloy, and a most surprising result was obtained. Here also 2 rows were lost. The 68 Cd cupels yielded 18.14 mg. Au, and the 34 without Cd yielded 9.28 mg. Au.

[2] *Engineering and Mining Journal, op. cit.*

This is 0.21 mg. more than half of the yield of the Cd cupels and shows a distinct protective action by the Cd.

Further evidence of this protective action of Cd is furnished by a test in which an alloy of Cd-Zn-Cu-Ag was used. The charge in the end cupels was 500+ mg. Au, 500 mg. alloy, 740 mg. Ag. This gave approximately 40 mg. each of Cd and Zn and 25 mg. Cu. The middle cupels held 500+ mg. Au, 1,125 mg. Ag, 20 mg. Cu. These charges were run in sets of 9, 3 rows of 3, with 4, 5, 8 and 10 grams of Pb. In every row except one, the middle cupel yielded more gold than either end cupel. In the exception, the middle cupel yielded 0.06 mg. more than one end, and 0.02 less than the other, or 0.02 mg. more than the average of the ends. The 24 end cupels yielded 10.42 mg. Au, and the 12 middle ones 5.89, which is 0.68 mg. more than half of the ends. The following table, with 4 Pb, is characteristic, all the cornets being plus:

| | First Row | | | Second Row | | | Third Row | | |
|---|---|---|---|---|---|---|---|---|---|
| Cornet | 0 45 | 0.30 | 0.30 | 0.39 | 0.28 | 0.33 | 0.34 | 0.18 | 0.35 |
| Cupel | 0.25 | 0.31 | 0.28 | 0.29 | 0.34 | 0 32 | 0.31 | 0.39 | 0.29 |

In some miscellaneous tests, 16 Cd end cupels yielded 3.58 mg. Au, and the 8 center ones, without Cd, 2.21 mg., 0.42 mg. more than half of the ends.

In many of the Cd cupellations, the characteristic dark brown oxide was shown.

*Copper.*—An alloy of 8 parts Ag, 2 parts Cu, designated as 800 Ag, was used. The assay consisted of 450+ mg. Au, and 1,250 mg. Ag-Cu alloy. They were run with 5 and 10 grams Pb at two temperatures, stated as high and low, in 3 rows of 3 cupels.

The figures in Table 4 call for further and more elaborate investigation, but time was lacking to continue this work.

TABLE 4 (First part)

| 5 Grams Pb | | | |
|---|---|---|---|
| Temp Low | | Temp. High | |
| Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. |
| 0.44− | 0.29 | 4.31− | |
| 0.13− | 0.26 | 4.19− | 3.66 |
| 0.28− | 0 32 | 4.22− | 3.81 |
| 0.62− | 0.37 | 2.16− | 1.23 |
| 1.00− | 0 26 | 2.02− | 1.93 |
| 0.88− | 0.27 | 1.58− | 1.74 |
| 0 98− | 0.43 | 1.54− | 1.48 |
| 1.16− | 0.49 | 1.69− | 1.70 |
| 1.23− | 0.66 | 1.67− | 1.89 |

TABLE 4 (Second part)

| 10 Grams Pb | | | |
|---|---|---|---|
| Temp. Low | | Temp. High | |
| Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg |
| 0 05+ | 0 25 | 2 47− | 2.89 |
| 0.04+ | 0 23 | 2.93− | 3.16 |
| 0.10− | 0.19 | 2 36− | 2.79 |
| 0 08− | 0.28 | 2.79− | 3.27 |
| 0.03+ | 0.32 | 3 15− | 3.55 |
| 0.18− | 0 29 | 3 31− | 3.57 |
| 0.14− | . | 3.20− | 3 56 |
| 0.01− | 0.32 | 3 17− | 3 51 |
| 0 91−* | 1.16* | 3 03− | 3 43 |

* Bead in cupel.

When Cu is present in bullion, we generally use more Pb and a higher temperature in cupellation. Both these conditions increase the cupel absorption of Au. In order to ascertain the effect of Cu *per se* upon the cupel absorption, 8 sets of 6 cupels, 2 rows of 3, were run with 5 to 7 mg. Cu, and 8 were run with 50 mg. Cu. Great care was exercised to have all of the other conditions as nearly as possible the same in every case, but a detailed inspection of the results indicates that the attempt was not altogether successful. The general temperature was shown by a pyrometer, and kept uniform, but the results show that the temperature of the beads did not conform to the pyrometer and varied considerably.

The cupel absorptions on the low Cu varied from 0.23 to 0.33 mg., and on the high Cu from 0.24 to 0.34 mg., but there was a very decided preponderance of low absorptions with low Cu and high ones with high Cu. The average absorptions were:

| | Mg. |
|---|---|
| High Cu | 0 2991 |
| Low Cu | 0.2773 |
| Difference | 0.0218 |

In order to keep the conditions the same, more Pb and a higher temperature were used on the low Cu, than were necessary, but the test clearly shows that Cu increases the cupel absorption.

The same fact is also shown in a general way by 26 rows with Cu in the ends, but without Cu in the centers. The Cu was alloyed with Ag and 2 proportions of Cu, 62.5 mg. and 125 mg. were each used with 4, 5, 8 and 10 grams Pb. The end cupels yielded 9.33 mg. Au, and the centers 4.12, or 0.545 mg., less than half of the ends.

*Lead.*—Being the universal constituent of cupelling beads, the action of this metal is of the utmost importance. Many results have been pub-

lished regarding its effects, but they are not, and could not really be expected to be, harmonious. There are so many variables involved that a tremendous amount of most painstaking work would be required to reach a quantitative expression of the relation of the weight of Pb in a bead to its effect upon the assay. I give here only a few illustrations, proposing to deal with the subject in detail in a separate paper on cupellation.

Under one set of conditions, 3 sets of 9 assays of fine gold showed the following cupel absorptions of Au, with differing amounts of Pb:

TABLE 5

| Pb, 0 5 Gram | | Pb, 2 Grams | | Pb, 5 Grams | |
|---|---|---|---|---|---|
| Mg. | Cupels | Mg. | Cupels | Mg. | Cupels |
| 0.06 | 2 | 0.20 | 1 | 0.53 | 1 |
| 0.07 | 4 | 0.21 | 4 | 0.54 | 1 |
| 0.10 | 1 | 0.22 | 3 | 0.55 | 2 |
| 0.12 | 1 | 0.26 | 1 | 0.56 | 2 |
| 0.15 | 1 | .... | ... | 0 58 | 1 |
| | | | | 0 60 | 1 |
| | 9 | | 9 | 0.64 | 1 |
| | | | | | 9 |

However, under a different set of conditions, and with 2 grams Pb, 8 sets of 6 assays of fine gold in one cupel gave 0.05 mg. absorption in one cupel, 0.06 mg. in 7 cupels, 0.07 mg. in 15 cupels, 0.08 mg. in 18 cupels, 0.09 mg. in 5 cupels and 0.11 in 2 cupels, total 48 cupels. Five sets of 6 assays in another cupel gave a preponderance of absorptions between 0.07 and 0.10 mg.

Even with 4 grams Pb, and on coin gold (900 Au, 100 Cu), the absorptions were less in another laboratory than they were on fine gold and 2 grams Pb in the first test.

Speaking in a very general way, under one set of conditions, doubling the Pb or increasing the pyrometer reading by 100° doubled the Au absorption, while both doubling the Pb and raising the pyrometer reading by 100° quadrupled the absorption of Au.

*Silver.*—Intimately connected with the action of base metals is the protective action of Ag upon the absorption of Au by the cupel. Here, again, much work is required to reach quantitative results, but the general effect is shown by the cupel absorption of Au in the base metal assay and the corresponding Au assay.

Various samples were taken from a mass melt weighing 774.3 oz., made from 33 bars. On these 16 Au and 16 base assays were made. The absorptions in the gold assays varied between 0.17 and 0.27 mg., with the

average 0.21 mg. The absorptions in the base assays varied between 0.64 and 1.00 mg. with the average 0.81 mg.

As a direct test of this question, sets of 6 cupels, 2 rows of 3, were run with varying proportions of Ag, and at 3 pyrometer readings. The total amount of gold absorbed in each set, and the average per cupel, were, in milligrams:

TABLE 6

| Parts of Ag | | Pyrometer Readings 900° C. | 950° C. | 1,000° C. |
|---|---|---|---|---|
| 3 | | 0.290 | 0.350 | 0.490 |
| | | 0.250 | 0.420 | 0.520 |
| | | 0.240 | 0.430 | 0.730 |
| | | 0.250 | 0.400 | 0 700 |
| | Av. per cupel...... ....... | 0.043 | 0.066 | 0 101 |
| 2½ | | 0.380 | 0.530 | 0.750 |
| | | 0.340 | 0.490 | 0.790 |
| | | 0.260 | 0.550 | 0.770 |
| | | 0.300 | 0.610 | 0.740 |
| | Av. per cupel...... . ..... | 0.053 | 0.091 | 0.127 |
| 2¼ | | 0.440 | ..... | 0 780 |
| | | 0.450 | ..... | 0.880 |
| | | 0.470 | ..... | 0.820 |
| | | 0 470 | ..... | 0.830 |
| | Av. per cupel...... ..... | 0 076 | ..... | 0.138 |
| 2 | | 0 570 | 0.620 | 0.920 |
| | | 0 520 | 0.660 | 0.830 |
| | | 0.630 | 0.930 | 0.970 |
| | | 0 700 | 0.820 | 0.930 |
| | Av. per cupel...... ... .... | 0.101 | 0.126 | 0 152 |

Under practical every-day conditions, the protection afforded by the Ag within ordinary parting conditions may be easily overbalanced by differences in other conditions.

I am greatly indebted to a former assistant, Mr. Scott Carter, for much painstaking effort in working out the data of this investigation.

*Conclusions*

In small amounts, Zn alone has no effect upon the cupel Au. In large amounts, the Zn-Au-Ag alloy is formed, comes to the surface and

carries Au to the cupel. A tendency to do this occurs when small amounts of Zn are cupelled with large amounts of Pb.

In all the comparative tests Cd showed a distinct protection of the Au, even when Zn was also present.

The presence of Cu *per se* directly increases the absorption of gold, but its effect is intensified by the higher temperature and larger amount of Pb required.

The effect of Pb varies greatly with the surrounding conditions.

The presence of Ag is a good protection to the Au, but may be overbalanced by other conditions.

*Addenda*

The following list of references on this subject has been prepared by the Library Service Bureau of the United Engineering Society. It was thought that these references were of sufficient interest to be included:

Rössler, H.: Investigations on the Assay of Gold as Applied to Gold Coins. *Dingler's Polytechn. Jour.* (1872), **206**, 188. Influence of lead on the cupellation of gold-bearing products.

Furman, H. van F.: Losses of Gold and Silver in the Fire-Assay. *Trans* (1894), **24**, 735. Amount of base metals in lead-button considered as one of the various conditions affecting losses during cupellation. Zinc appears to be the most objectionable element. Copper also increases the losses.

Furman, H. van F.: *A Manual of Practical Assaying*, 1903, 5th ed., New York, pp. 386–400. Losses of gold and silver in the fire-assay.

Lodge, R. W.: *Notes on Assaying and Metallurgical Laboratory Experiments*, 1904, New York, John Wiley & Sons, 143–144. Table showing effect of copper on cupellation of gold, with lead, gold and temperature remaining constant.

Rose, T. K.: On Cupellation and Parting in Ore Assaying *Jour. Chem. Met. Soc. South Africa* (1905, January), **5**, 167. Result of some experiments on the losses in cupellation due to the presence of various impurities in the lead buttons are given in tabulated form. In each case the charge consisted of 1 mg. of gold, 4 mg. of silver and 1 gram. of the impurity with 25 grams of lead.

Fulton, Ch. H.: *A Manual of Fire Assaying*, 1907, Hill Publishing Co., pp. 71–80, particularly Table XIV. Copper in cupellation of silver and gold, pp. 121–123. Losses in assay for gold and silver due to amount of lead with which gold and silver is cupelled, and the nature and amount of impurities present.

Smith, E. A.: Assay of Industrial Gold Alloys. *Jour. Inst. Metals* (1910), **3**, 98–123. Effect of various properties of lead for cupellation of gold-copper alloys extensively discussed. Results of experiments given in tabulated form

Smith, E. A.: *Influence of Base Metals on Cupellation.* Sampling and assay of the precious metals: comprising gold, silver, platinum, and the platinum group metals in ores, bullion and products. London, 1913, Ch. Griffin, pp. 167–176. Complete critical review of the literature on this subject, together with some results obtained by the author himself (Table XVII, p. 173).

Dewey, F. P.: Determination of Silver and Base Metal in Precious Metal Bullion. *Jour. Ind. Engng. Chem.* (1914), **6**, 650–658; 728–736.

Rose, T. K.: *The Metallurgy of Gold*, 6th ed., London, 1915, pp. 507–509. Influences of base metals on cupellation.

# Graphic Solutions of Some Compressed-air Calculations

BY C. W. CRISPELL,* NEW HAVEN, CONN.

(St. Louis Meeting, October, 1917)

THE four nomograms presented in this article were designed to simplify and make more rapid the calculations connected with the compression and transmission of air. The formulæ involved are rather complicated, but the graphic solutions are simple; their use requires an ability to follow explicit directions and to interpolate values on a logarithmic scale.†

For the sake of convenience, the nomograms will be divided into two classes, and the purpose and accuracy of each will be discussed.

## POWER FOR COMPRESSION

It is frequently desirable to determine the horsepower necessary to compress a given amount of free air from atmospheric pressure to a certain final pressure. The nomograms in Fig. 1 and 2 present a rapid method of calculating the horsepower required to compress 1 cu. ft. of free air per minute by single or two-stage compression. The general formula for the horsepower required to compress air adiabatically is:

$$HP = \frac{S \times 144}{33{,}000} \cdot \frac{PVn}{(n-1)} \left[ \left( \frac{P'}{P} \right)^{\frac{n-1}{Sn}} - 1 \right] \quad . \; . \; . \; . \; . \quad (1)$$

where:

$P$ = absolute atmospheric pressure at compressor, in pounds per square inch.

$P'$ = final absolute pressure of compressed air, in pounds per square inch.

$V$ = volume of free air compressed per minute, in cubic feet.

$S$ = number of stages.

$n$ = exponent of the compression curve.

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† In order to meet publication requirements, the nomograms have been reproduced on a reduced scale and are intended merely to illustrate the method of determining compressed-air data, but separate large-scale plates have been prepared, suitable for working charts, which members may obtain free of charge by addressing the Secretary.

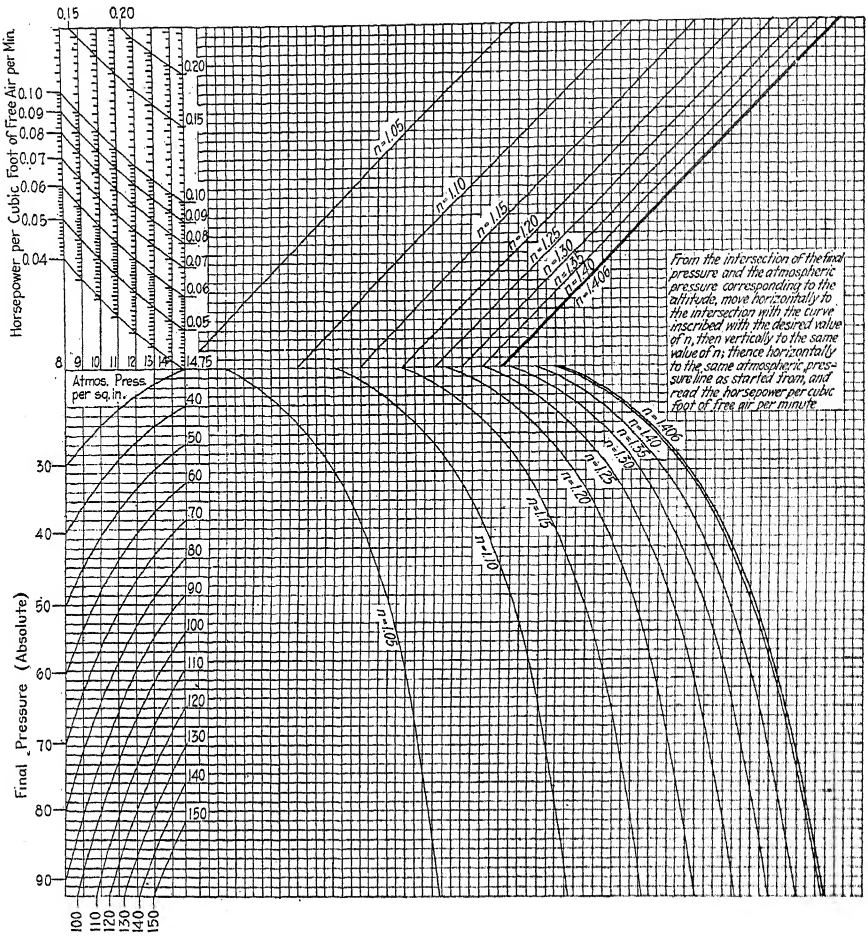


FIG. 1.

| Altitude in Feet | Barometric Pressure | | Altitude in Feet | Barometric Pressure | |
|---|---|---|---|---|---|
| | Inches, Mercury | Pounds per Sq. In. | | Inches, Mercury | Pounds per Sq. In. |
| 0 | 30.00 | 14.75 | 8,000 | 22.11 | 10.87 |
| 1,000 | 28.80 | 14.20 | 9,000 | 21.29 | 10.46 |
| 2,000 | 27.80 | 13.67 | 10,000 | 20.49 | 10.07 |
| 3,000 | 26.76 | 13.16 | 11,000 | 19.72 | 9.70 |
| 4,000 | 25.76 | 12.67 | 12,000 | 18.98 | 9.34 |
| 5,000 | 24.79 | 12.20 | 13,000 | 18.27 | 8.98 |
| 6,000 | 23.86 | 11.73 | 14,000 | 17.59 | 8.65 |
| 7,000 | 22.97 | 11.30 | 15,000 | 16.93 | 8.32 |

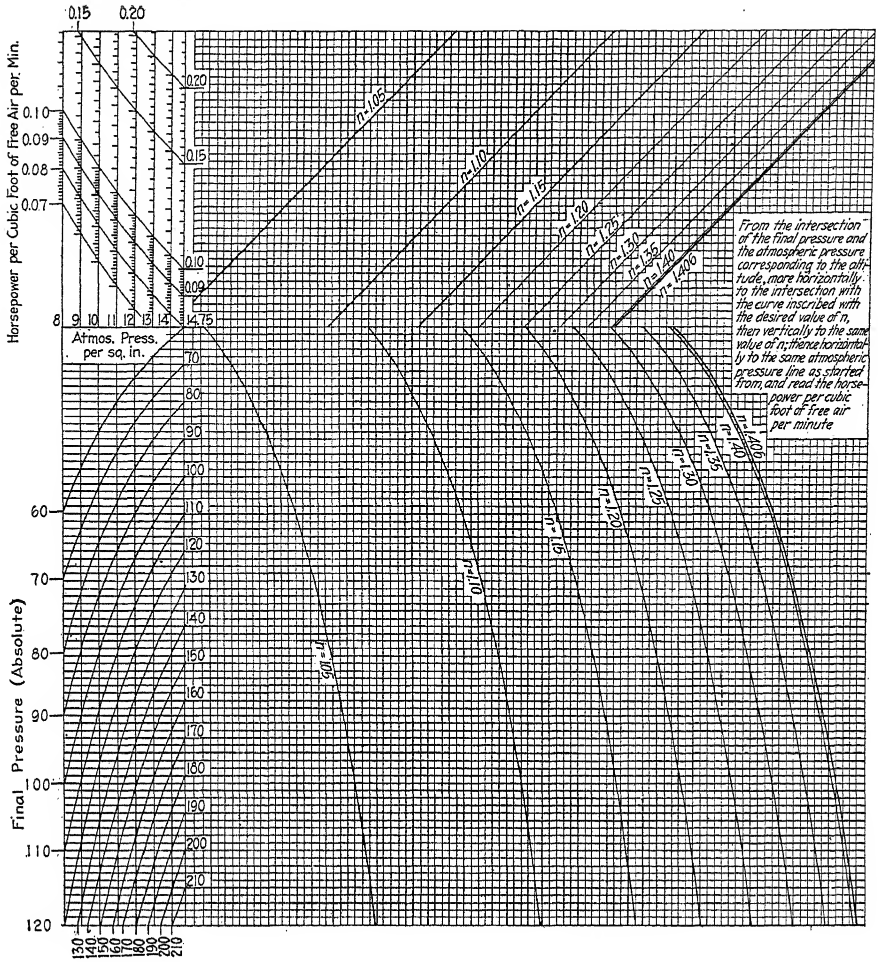


FIG. 2.

| Altitude in Feet | Barometric Pressure | | Altitude in Feet | Barometric Pressure | |
|---|---|---|---|---|---|
| | Inches, Mercury | Pounds per Sq. In. | | Inches, Mercury | Pounds per Sq. In. |
| 0 | 30.00 | 14.75 | 8,000 | 22.11 | 10.87 |
| 1,000 | 28.80 | 14.20 | 9,000 | 21.29 | 10.46 |
| 2,000 | 27.80 | 13.67 | 10,000 | 20.49 | 10.07 |
| 3,000 | 26.76 | 13.16 | 11,000 | 19.72 | 9.70 |
| 4,000 | 25.76 | 12.67 | 12,000 | 18.98 | 9.34 |
| 5,000 | 24.79 | 12.20 | 13,000 | 18.27 | 8.98 |
| 6,000 | 23.86 | 11.73 | 14,000 | 17.59 | 8.65 |
| 7,000 | 22.97 | 11.30 | 15,000 | 16.93 | 8.32 |

The exponent $n$ is the ratio between the specific heat of air at constant pressure and its specific heat at constant volume. For theoretical adiabatic compression $n = 1.406$. The value of $n$ in actual compression varies with the design of the compressor and the efficiency of the cooling arrangements. In well designed single-stage dry compressors, the value of $n$ ordinarily lies between 1.25 and 1.35. Well designed two-stage compressors give values of $n$ as low as 1.15, though 1.20 to 1.25 is more common.[1] Additional values of $n$ have been added on the nomograms for the sake of completeness. Values of $P$ for different altitudes are given in the tables accompanying Fig. 1 and 2.

Directions for using Fig. 1 and 2 are printed on them. It should be noted that the scale for the horsepower per cubic foot of air starts from a different ordinate for each value of $P$. Formula (1), of course, makes no allowance for the mechanical losses in the compressor.

### *Speed and Accuracy*

To test the diagrams, six problems have been taken at random as shown in Tables 1 and 2. They were solved first by the use of Fig. 1

TABLE 1.—*Single-Stage Compression*

| Problem No. | 1 | 2 | 3 | 4 | 5 | 6 |
|---|---|---|---|---|---|---|
| Final pressure | 70.0 | 60.0 | 90.0 | 80.0 | 50.0 | 40.0 |
| $n$ | 1.30 | 1.35 | 1.40 | 1.25 | 1.15 | 1.20 |
| Atmos. pressure | 14.7 | 13.0 | 12.0 | 11.0 | 10.0 | 8.0 |
| Horsepower: | | | | | | |
| By chart | 0.122 | 0.108 | 0 144 | 0.118 | 0.079 | 0.066 |
| By slide rule | 0.121 | 0.107 | 0.143 | 0.116 | 0.078 | 0.065 |
| By logs | 0.120 | 0.107 | 0.143 | 0.117 | 0.078 | 0.065 |
| Per cent. error: | | | | | | |
| By chart | +1.60 | +0.93 | +0.70 | +0.85 | +1.28 | +1.54 |
| By slide rule | +0.80 | 0.0 | 0.0 | −0.85 | 0.0 | 0.0 |

TABLE 2.—*Two-Stage Compression*

| Problem No. | 1 | 2 | 3 | 4 | 5 | 6 |
|---|---|---|---|---|---|---|
| Final pressure | 150.0 | 130.0 | 110.0 | 100.0 | 80.0 | .60.0 |
| $n$ | 1.40 | 1.35 | 1.30 | 1.25 | 1.20 | 1.15 |
| Atmos. pressure | 14.7 | 14.0 | 13.0 | 12.0 | 10.0 | 9.0 |
| Horsepower: | | | | | | |
| By chart | 0.177 | 0.159 | 0.139 | 0.125 | 0.100 | 0.080 |
| By slide rule | 0.175 | 0.158 | 0.138 | 0.125 | 0.099 | 0.079 |
| By logs | 0.175 | 0.158 | 0.138 | 0.124 | 0.099 | 0.079 |
| Per cent. error: | | | | | | |
| By chart | +1.14 | +0.63 | +0.72 | +0.80 | +1.01 | +1.26 |
| By slide rule | 0.0 | 0.0 | 0.0 | +0.80 | 0.0 | 0.0 |
| Mean error by chart | 0.93% | | | | | |
| Mean error by rule | 0.13% | | | | | |

[1] R. Peele: *Compressed-Air Plant*, Wiley, N. Y., 1913.

and 2 and then by means of a 20-in. slide rule applied to formula (1). The average time required to solve a problem with the diagrams was 1 min. 45 sec. as compared to 3 min. 45 sec. for the slide rule. The mean error in these calculations, disregarding + and − values, was 1.15 per cent. for the graphic solution and 0.28 per cent. for the slide rule.

## Transmission of Compressed Air

Fig. 3 gives a graphic solution of D'Arcy's formula for the transmission of compressed air in pipes. This formula is:[2]

$$D = c\sqrt{\frac{d^5 (p_1 - p_2)}{w_1 l}} = \frac{c\sqrt{d^5}}{\sqrt{l}}\sqrt{\frac{(p_1 - p_2)}{w_1}} \quad . \; . \; . \; . \; . \qquad (2)$$

where:

$D$ = the volume of compressed air in cubic feet per minute discharged at the final pressure.

$c$ = a coefficient varying with the diameter of the pipe, as determined by experiment.

$d$ = actual diameter of pipe in inches.

$l$ = length of pipe in feet.

$p_1$ = initial gage pressure in pounds per square inch.

$p_2$ = final gage pressure in pounds per square inch.

$w_1$ = the density of the air, or its weight per cubic foot, at initial pressure.

The nomogram, Fig. 3, allows the solution of this equation without reference to tables for values of $c\sqrt{d^5}$ and $\sqrt{\frac{p_1 - p_2}{w_1}}$ which are ordinarily used for this purpose.

The following cases may arise in connection with the transmission of compressed air through pipes:

1. Given the volume of compressed air, the length of pipe, the initial pressure, and the maximum allowable drop in pressure; required, the diameter of the pipe.

2. Given the length of pipe, the diameter of pipe, the initial pressure, and the maximum allowable drop in pressure; required, the volume of compressed air which the pipe will carry.

3. Given the diameter of the pipe, the volume of compressed air, the initial pressure, and the maximum allowable drop in pressure; required, the maximum length of pipe of the given diameter which can be used under these conditions.

4. Given the diameter of the pipe, the volume of compressed air, the length of pipe, and the maximum allowable drop in pressure; required, the pressure at which the air must enter the pipe.

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[2] R. Peele: *Op. cit.*

5. Given the diameter of the pipe, the volume of compressed air, the length of pipe, and the initial pressure; required, the drop in pressure.

The graphic solution of each of these problems may be obtained from Fig. 3 as follows:

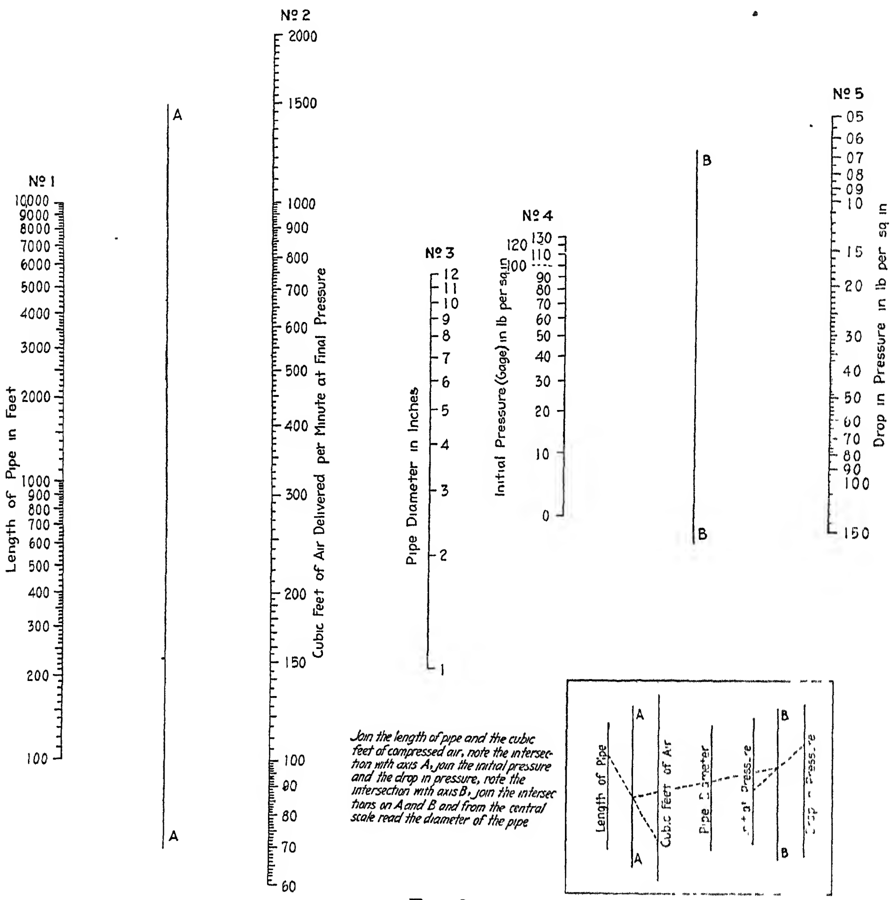


FIG. 3.

*Case* 1.—With a straight edge, join the length of pipe and the cubic feet of compressed air (not free air); note the intersection on axis $A$; join the initial pressure with the drop in pressure; note the intersection on axis $B$. A line joining the two points on $A$ and $B$ will intersect scale number 3 at the required pipe diameter.

*Case* 2.—With a straight edge, join the initial pressure with the drop in pressure, note the intersection on axis $B$; join the intersection on $B$ with the diameter of the pipe; note the intersection on axis $A$. A line joining this point on $A$ with the length of pipe will intersect scale number 2 at the required volume of compressed air.

*Case* 3.—With a straight edge, join the initial pressure with the drop in pressure; note the intersection on axis $B$; join the intersection on $B$ with the diameter of the pipe; note the intersection on axis $A$. A line joining this point on $A$ with the given cubic feet of compressed air intersects scale number 1 at the required maximum length of pipe.

*Case* 4.—With a straight edge, join the length of pipe with the volume of compressed air; note the intersection on axis $A$; join the intersection on $A$ with the diameter of the pipe; note the intersection on axis $B$. A line joining this point on $B$ with the allowable drop in pressure intersects scale number 4 at the required initial pressure at the entrance of the pipe.

*Case* 5.—With a straight edge, join the length of pipe with the volume of compressed air; note the intersection on axis $A$; join the intersection on $A$ with the diameter of the pipe; note the intersection on axis $B$. A line joining this point on $B$ with the given initial pressure will intersect scale number 5 at the required drop in pressure.

### *Speed and Accuracy*

The six problems given in Table 3 were worked out to test this nomogram. An average time of 1 min. 45 sec. was required to solve a problem using a 20-in. slide rule in connection with the tables published in Peele's *Compressed-Air Plant.* These same problems were solved by the nomogram in an average time of 1 min. The mean error of the nomogram is less than 0.5 per cent. as shown by Table 3.

TABLE 3.—*D'Arcy's Formula*

| | | | | | | |
|---|---|---|---|---|---|---|
| Problem No. | 1 | 2 | 3 | 4 | 5 | 6 |
| Size of pipe. | 2 | 4 | 6 | 8 | 10 | 12 |
| Length of pipe | 100 | 1,000 | 2,000 | 3,000 | 4,000 | 5,000 |
| Initial pressure. | 40 | 60 | 70 | 80 | 90 | 100 |
| Drop in pressure | 2.0 | 2.0 | 3.0 | 4.0 | 5.0 | 8.0 |
| Cubic feet of air: | | | | | | |
| By chart | 78.2 | 133.0 | 309.0 | 574.0 | 930.0 | 1,600.0 |
| By slide rule. | 78.8 | 133.6 | 308.2 | 573.0 | 935.0 | 1,603.0 |
| By logs | 78.9 | 133.4 | 308.3 | 573.0 | 935.0 | 1,603.0 |
| Per cent. error: | | | | | | |
| By chart | −0 88 | −0.30 | +0.22 | +0.17 | −0.53 | −0.18 |
| By slide rule | −0.12 | +0.15 | −0 03 | 0.0 | 0.0 | 0.0 |

## PRESSURE-VOLUME RELATIONSHIPS

Many of the data on the capacity of air compressors and the air consumption of rock drills or other machines are expressed in terms of cubic feet of free air. The volume of compressed air to be transmitted must be known before the nomogram for D'Arcy's formula, Fig. 3, can be used. Volumes of free air at different altitudes can be converted into corre-

sponding volumes of compressed air at different absolute pressures by means of nomogram on Fig. 4. This diagram is based on the equation:

$$pv = p'v'$$

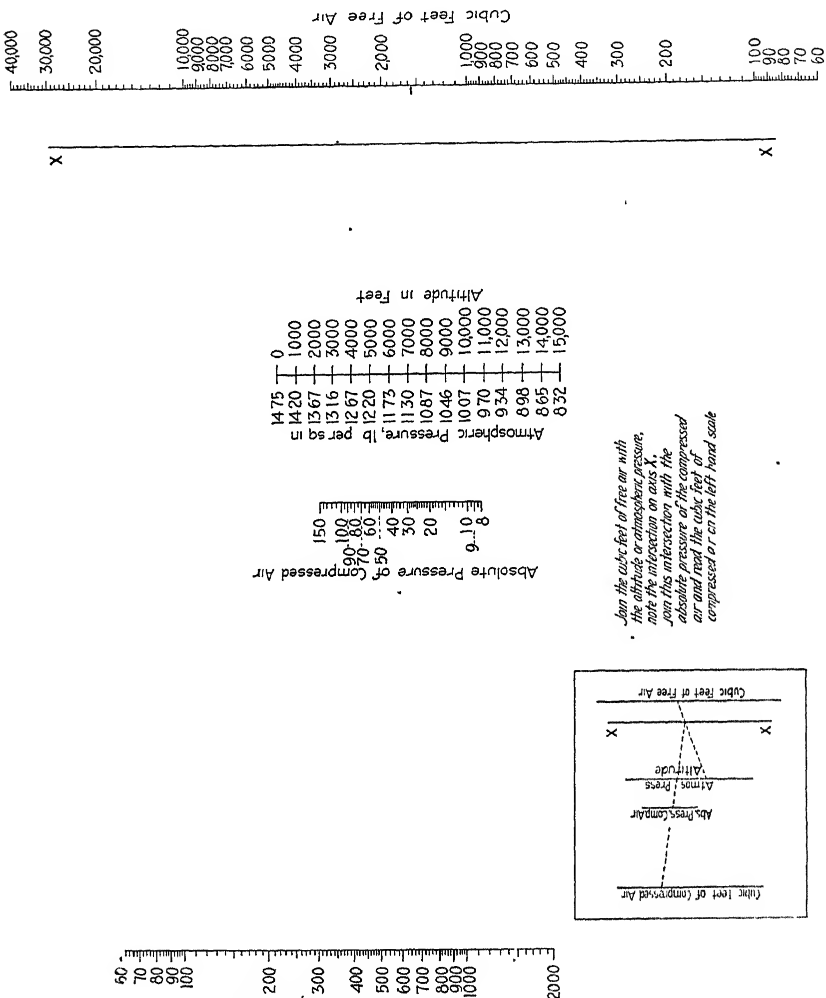

FIG. 4.

where:

$p$ = initial absolute (atmospheric) pressure in pounds per square inch.

$v$ = initial volume of air in cubic feet.

$p'$ = final absolute pressure of compressed air in pounds per square inch.

$v'$ = volume of compressed air in cubic feet.

The accuracy of this chart as compared to calculations with a 20-in. slide rule and with logarithms is shown by Table 4.

TABLE 4

| | | | | | | |
|---|---|---|---|---|---|---|
| Problem No. | 1 | 2 | 3 | 4 | 5 | 6 |
| Altitude. | 0 | 1,000 | 2,000 | 3,000 | 6,000 | 14,000 |
| Press. of comp. air. | 20 | 40 | 60 | 80 | 90 | 100 |
| Cubic feet of free air. | 100 | 500 | 1,000 | 1,500 | 2,000 | 3,000 |
| Cubic feet of comp. air: | | | | | | |
| By chart. | .74.0 | 178.0 | 227.0 | 248.0 | 260.0 | 259.0 |
| By slide rule. | 73.8 | 177.5 | 228.0 | 247.0 | 261.0 | 259.5 |
| By logs. | 73.8 | 177.5 | 227.8 | 246.8 | 260.7 | 259 5 |
| Per cent. error: | | | | | | |
| By chart | +0.27 | +0 28 | −0.35 | +0.49 | −0.26 | −0.15 |
| By slide rule. | 0.0 | 0.0 | +0.09 | +0.08 | +0.11 | 0.0 |

The figures given above for speed of work with Fig. 1, 2, and 3, as compared with the slide-rule work, do not indicate the true value of the nomograms. The problems were worked out at a time when the author had been using equations (1) and (2) and was entirely familiar with their terms and their algebraic solution. At a later date, when he had become somewhat rusty perhaps both in his mathematics and his knowledge of compressed air, the graphic solution would take relatively much less time and be more apt to give numerically correct results.

In conclusion, the author desires to give credit to Prof. H. L. Seward for many helpful suggestions regarding the construction of the nomograms and to Prof. J. F. McClelland for kind assistance in the preparation of this article.

# Methods for Determining the Capacities of Slime-thickening Tanks

BY R. T. MISHLER,* ESQUEDA, SONORA, MEX.

(St. Louis Meeting, September, 1917)

I WISH to express my keen appreciation of the article on the above subject by Coe and Clevenger.[1] It has been doubly interesting to me, for the reason that the experience recorded and the principles evolved practically parallel the results of similar investigations made by myself and others at The Lucky Tiger Mine, El Tigre, Sonora, Mex. An outline of the work at El Tigre may aid in confirming the principles expressed by Coe and Clevenger.

El Tigre has the unenviable distinction of possessing a slime with a much lower settling rate than any recorded in the article discussed. Under most favorable conditions the settling rate is about one-third that reported for the Liberty Bell mill, and one-tenth that reported for the Nipissing, Golden Cycle, and Presidio mills. This slow settling characteristic of Tigre ore has necessitated careful investigation of settling phenomena throughout the various evolutions of the plant.

## Previous Settling Investigation at Tigre

During construction work at the mill, the metallurgical engineer in charge, D. L. H. Forbes, ran a series of experiments on settling. His tests proved that deep settling tanks were not ordinarily necessary. The use of tanks 2 or 3 ft. in depth was suggested.[2]

Later experiments demonstrated that the following principles of settling applied to Tigre pulp.[3]

1. In settling dilute pulp, settling rates are independent of pulp depth.

2. In settling thick pulp, settling rates are approximately proportional to pulp depth.[4]

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[1] H. S. Coe and G. H. Clevenger: *Trans.* (1916), **55**, 356.

[2] D. L. H. Forbes: Settling of Mill Slimes, *Engineering and Mining Journal* (Feb. 24, 1912), **93**, 411.

[3] R. T. Mishler: Settling Slimes at the Tigre Mill, *Engineering and Mining Journal* (Oct. 5, 1912), **94**, 643.

[4] Coe and Clevenger express this principle in the words: "After pulp reaches the consistency where the flocs touch each other, further elimination of water becomes approximately a function of time."

3. The critical dilution, or the boundary between dilute and thick pulp, in the application of the above principles, is the highest dilution at which channels form in the pulp; clear liquid reaching the surface through the channels.

It is gratifying to note that these principles, tentatively announced 4 years ago as applying to Tigre ore, are now proved, by the comprehensive tests of Coe and Clevenger, to have universal application.

The formula, suggested at that time for solving problems of area, was likewise similar to that adopted by these gentlemen.

$$\text{Area} = \frac{0.0222\,(\text{Dry Tons})\,(R_1 - R_2)}{\text{Settling Rate, in Feet per Minute}}.$$

$R_1$ and $R_2$ were the moisture ratios respectively of feed and discharge. The settling rate was that of the feed, as determined by laboratory tests. This method of calculation was shown to give results which closely approximated those of actual settling operations in the plant.

In subsequent investigations at El Tigre, improved methods have been devised for the laboratory determination of settling rates, and for the calculations of the required area and depth of settling tanks. Also many interesting new principles have been developed concerning the effect on settling phenomena of various changes in temperature and percentage of lime in solution. The details of these investigations are discussed at length below.

## Definitions

In order to avoid confusion, the nomenclature employed by Coe and Clevenger is used as far as possible in this paper.

Settling rate is expressed in terms of the depth in feet of clear liquid formed per hour; or the rate in feet per hour at which the pulp surface moves downward.

Dilution is expressed in terms of the ratio by weight of liquid to solids, and is termed, for short, "The L : S Ratio." In tables, charts and formulas, the first factor only of the L : S ratio is used. Thus the figure 5 designates a dilution of 5 : 1.

Natural slime is defined as minutely divided particles of pulp resulting from the disintegration of kaolin, talc, or other similar constituents of ore which occur minutely subdivided in nature.

Slime is empirically defined, for the purpose of this discussion, as the finer particles of pulp that will remain suspended for 5 min. in a 100 : 1 mixture of fresh water and ground ore; the temperature of separation being 60° F. (16° C.). The term covers all the natural slime occurring in the pulp and also a considerable portion of extremely fine sand.

Sand[5] is the relatively coarse material of the pulp, resulting from grinding quartz, calcite, feldspar, or other massive gangue matter. It may vary from the size of a pea to impalpable powder.

Flocs or floccules are aggregates of slime particles, caused by attraction or cohesion between the particles. Flocculation[6] may be increased by the addition of electrolytes, such as lime hydrate, magnesium sulphate, glue, etc., and may be decreased by the introduction of deflocculating agents (sodium carbonate, sodium hydrate, etc.). Temperature and dilution have marked effects on the rate at which flocs form.

The free-settling zone is the horizon of evenly flocculated dilute pulp in the upper part of a continuous-settling tank. In this zone, the flocs do not press one upon the other, and settling rates are independent of pulp depth.

The critical dilution is the dilution at which flocs begin to touch. It is the highest dilution at which channels or tubes form in the pulp, the channels or tubes furnishing passageways to the surface for the clear liquid set free in the bottom of the vessels.

The thickening zone is the zone of thick pulp in the bottom of a continuous-settling tank. In this zone, the flocs rest one upon the other, and liquid is eliminated chiefly through the channels mentioned above. Settling rates, in the thickening zone, are approximately proportional to pulp depth. The same principle is expressed by Coe and Clevenger in the words: "Elimination of water is a function of time" (in the thickening zone). The term, "zone of compressions" is used by Coe and Clevenger, on the assumption that the introduction of the time element, as a function of settling, is due to compression caused by the depth of pulp. I believe that this assumption is open to dispute and have hence adopted a more general term for designating the zone.

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[5] It has been demonstrated by Free, *Engineering and Mining Journal* (Nov. 6, 1916), **101**, 250, that as regards the relation of mineral particles to liquid, there is no dissimilarity in the physical characteristics of slime and sand; any variation in behavior being explained on the basis of the difference in size of the particles, and the consequent effect of surface tension, surface adsorption and other similar forces. Free also shows that it is doubtful if sand is ever ground sufficiently fine, in ordinary mill practice, to cause it to assume the colloidal properties of natural slime. On this account it is necessary for the hydrometallurgist to distinguish between sand and natural slime. Regardless of its degree of fineness, sand has little effect on settling phenomena. It is the natural slime in the ore which determines settling rate, critical dilution and other phases of settling. It is likewise the natural slime in ore which determines the rate of filtration and the dilution best suited for classification, flotation, or gravity concentration.

[6] This phase of settling is well covered by Free, and also by Ralston, *Engineering and Mining Journal* (May 20, 1916), **101**, 894.

## Theoretical Considerations

Investigations at El Tigre indicate that cohesion or attraction between the flocs is responsible for the fact that elimination of water in the thickening zone is a function of time. Only a positive tensile force, like cohesion, could form channels and keep them open afterward. In the absence of channels, elimination of water in the thickening zone is a function of area and not of time. This fact is illustrated in Table 3.[7]

Cohesion plays an important part during all settling operations. In the free-settling zone it causes the slime particles to aggregate into flocs. In the first stages of thickening, it causes the flocs to draw together, thus developing tensile strains in the pulp which are relieved by cracks and channels (the clear liquid reaching surface through these avenues). During the final stages of thickening, especially in the presence of high lime, cohesion no longer causes closer union between slime particles, or between flocs; but it still strongly resists any force tending to change the shape of the flocs.[8] This resistance reduces settling rate by opposing the compression, due to pulp depth, which would otherwise force the flocs to displace the liquid in the interfloccular spaces. The greater density attained by the deeper pulp columns in Table 3 can best be explained on this basis.

It has been noted by Coe and Clevenger that after settling has ceased in the shorter columns, it may be caused to continue by decanting the clear liquid and agitating the thick pulp. This behavior is quite as apparent in the absence of classified sand as when classified sand occurs in the pulp. The cause of the action appears to be the rearrangement of the slime particles. Before agitation, a slightly flocculated structure is noted, a close inspection disclosing clear liquid between the flocs. After agitation, no flocculation is apparent. Evidently the pulp flocculates "en masse" after agitation, and settling is resumed because cohesion aids instead of resists compression.

The rate at which flocs form in the free-settling zone is an item of importance in the operation of continuous-settling tanks. When pulp first leaves the feed column of a tank, flocs are entirely unformed and the settling rates are low or nil. The dilute pulp spreads out laterally—over the top of the denser pulp in the bottom of the tank. As the lateral

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[7] The failure of the channels in the latter part of the tests of Table 3 was due to high lime, combined with the complete absence of classified sand. This combination of conditions seldom occurs in plant practice. Hence the data of Table 3 have little practical value, except to aid in understanding the principles of settling. Classified sand, when it occurs in the pulp, furnishes passageways for the liquid after the collapse of the channels. Under such conditions, settling rates remain proportional to depth throughout the entire thickening period.

[8] It is in the stage of thickening that the channels collapse (see Table 3).

movement lessens, flocs begin to form and the settling rate increases correspondingly. The rate at which flocs develop depends upon temperature, dilution, and strength of electrolytes. During warm weather, and in the presence of high lime, the slime becomes fully flocculated almost immediately after leaving the feed column. Under such conditions the average settling rate, over the entire area of the tank, is high. During cold weather, and in the presence of low lime, partially flocculated pulp may spread practically to the tank periphery, forming a blanket over the pulp surface, which retards egress of liquid from the more completely flocculated pulp in the lower horizons of the tank.

The various stages in the development of flocs are well represented in laboratory tests at free-settling dilutions. When tests are first started, flocculation is imperfect and settling rates are low, this stage representing conditions near the feed column in continuous-settling tanks. As the tests continue, flocs become more perfectly developed and settling rates increase. When lime, dilution, and temperature are high, flocculation is completed and maximum settling rates are attained before the pulp surface has settled ⅛ in. When lime, dilution and temperature are low, complete flocculation and maximum settling rate are attained only after the pulp surface has settled ¼ or ½ in. The complete flocculation and consequent maximum settling rates, finally attained in the tests, correspond to the conditions existing near the tank periphery and also in the lower free-settling horizons of the tank.

Laboratory tests, for determining the settling rates corresponding to the various horizons in a continuous-settling tank, should be so performed that the degree of flocculation in the tests is approximately the same as occurs at the various depths in the settling tanks. Tests at the dilution of feed should aim to represent the imperfectly flocculated condition occurring near the feed column as well as the more complete flocculation at the tank periphery, the average settling rate over the entire pulp surface being the result desired. Experience at Tigre, under varying conditions of dilution, lime and temperature, indicates that this result can best be accomplished when the first 0.05 ft. settled in the tests furnishes the basis for the calculation of the settling rates corresponding to feed dilutions.

In case doubt exists that area might be governed by the permeability of some horizon below that of the feed, a series of settling tests at lower free-settling dilutions are run. In this series, observations are not started until after flocs are fully developed and maximum settling rates are attained. This condition is usually fulfilled when the second 0.05 ft. settled in the tests furnishes the basis for calculating the settling rates corresponding to the lower dilutions in the depths of the settling tank.[9]

[9] In order to maintain uniform dilution at the pulp surface, the depth of pulp in the tests should be 1 ft. or more.

At Tigre, area is invariably governed by the settling rate of the feed horizon. Hence, the tests at lower dilutions are usually omitted. In Table 2, a series of settling rates, experimentally determined at feed dilutions, are compared with actual settling rates in a 32-ft. continuous-settling tank, a considerable range in lime, dilution and temperature being obtained in the comparison. The experimentally determined settling rates will be observed to check closely with those occurring in actual practice. Minor differences are probably due to changes in the character of ore from day to day, and to the difficulty of properly correlating dilutions of feed and discharge. The tank was operated at maximum capacity throughout the tests, so errors introduced by changes in pulp level were slight.

Table 3 also illustrates the effect of deficient area on thickening operations below critical dilution. In all cases recorded in the table, the thickness of discharge was limited by settling rates in the free-settling zone, the area being inadequate to secure any benefit from the depth of tank. Even when the feed entered the tank below critical dilution, it was diluted to critical dilution by the excess of liquid rising from the lower depths, the bulk of pulp in the tank being practically at critical dilution. Final thickness in such cases was limited by the settling rate at critical dilution.[10]

## Practical Principles

The foregoing discussion of settling phenomena differs somewhat from that of Coe and Clevenger. The differences are mostly theoretical in character and are not insurmountable in a practical consideration of the subject. For the application of experimental data to the solution of practical problems, fundamental principles are all-important. In the conception of fundamental principles, Coe, Clevenger and myself are fully agreed.

The principles underlying settling operations may be briefly stated as follows:

1. Settling rates in the free-settling zone are independent of pulp depth. Stated in other words: At dilutions above critical dilution, elimination of water is a function of settling area, and is independent of pulp depth.

2. Settling rates in the thickening zone vary directly with pulp depth. Stated in other words: At dilutions below critical dilution, elimination of liquid is a function of pulp volume, being dependent on both area and depth. Or, in the words of Coe and Clevenger, "elimination of water in the zone of compression is a function of time."

[10] Settling rates, corresponding to dilutions of feed below critical dilution, were regarded as proportional to differences in dilution, the calculations being based on the experimentally determined settling rate at critical dilution.

3. Settling rates vary with the dilution of the natural slime in the pulp, being usually greatest for high dilutions and smallest for low dilutions.

4. Settling rates are practically independent of the amount of sand occurring in the pulp (regardless of the fineness of the sand).

5. Settling rates increase as the temperature increases.[11]

6. Settling rates vary with the degree of flocculation of the slime particles.

7. Degree of flocculation depends upon:

(*a*) The nature and preceding treatment of the ore.

(*b*) The character and concentration of electrolyte.

(*c*) The dilution and temperature of the pulp.

(*d*) The time elapsed after inception of flocculation.

8. Critical dilution[12] depends primarily upon the character and proportion of natural slime in the ore; secondarily upon the strength of flocculating agents in solution; and thirdly (only slightly) upon temperature.

The above principles furnish the basis for the solution of settling problems. The first two principles are of direct importance in the determinations, by laboratory tests, of the settling equipment required in metallurgical plants. The remaining principles enumerate the effects of other influences upon settling operations and illustrate the importance of maintaining these influences the same in the tests as in actual practice.

## Tests for Determining Settling Equipment

In designing or remodeling hydro-metallurgical plants, questions frequently arise regarding the number and size of settling tanks to be installed. Occasionally the tanks are required for some definite service in a predetermined flow sheet of the plant. In such cases, the conditions of temperature, electrolyte, tonnage, dilution of feed, and desired thickness of discharge, are known beforehand, the problem being to provide sufficient settling equipment to fulfil these conditions. Usually, the flow sheet depends to some extent upon settling operations, rendering it necessary to determine approximately the settling equipment required under varying conditions, in order to decide intelligently upon the conditions best suited for a well balanced plant.

The first step, in the solution of such problems, is to secure a repre-

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[11] This phase of settling is covered by Ashley, *Mining and Scientific Press* (June 12, 1909), **98**, 831.

[12] The critical dilution, in ordinary "all-slime" plants, is usually about 2 : 1 when lime is low and 3 : 1 when lime is high. Tigre ore contains an unusually large proportion of natural slime. The critical dilution is therefore high, ranging from 4 : 1 with low lime to 6 : 1 with high lime. The critical dilution of Tigre slime (with all possible sand removed by classification) varies between 7 : 1 and 12 : 1 according to the lime content.

sentative sample of the pulp to be settled. The sample should contain the maximum proportion of natural slime that would occur in the mill pulp during any single day of operation. It should secure as nearly as possible the same preliminary treatment as would occur in mill practice. Above all, the sample should not be dried at a high heat, since heat changes the character of the slime, greatly increasing subsequent settling rates. The importance has been pointed out, by Coe and Clevenger, of not allowing the sample to remain a greater time in contact with water than would occur in plant operations; the degree of flocculation, and consequently the settling rate, being changed by this treatment. The electrolytes present in the water should be the same as would occur in plant practice. Often, electrolytes are derived partly from the ore[13] and can be introduced in proper proportions only by bringing the water into repeated contact with fresh samples of ore, the procedure being the same as in the return of water to the head of the mill. The entire object of these precautions is to secure a sample that will represent the pulp to be fed to the tanks.

This sample is settled, preferably over night. The clear water is decanted and set aside for diluting the pulp in the various tests. The thickened pulp is mixed, and tested for liquid to solid ratio.

The following procedure in testing has given most uniformly reliable results at Tigre:

For determining critical dilution, glass graduates are filled to a depth of 1 ft. with varying consistencies of the pulp to be tested. After 10 or 15 min. of undisturbed settling, channels develop in the columns of thicker pulp, clear liquid rising through the channels. If no channels are observed, or if they develop first in the bottom of the vessel, the critical dilution is lower than that of the test. If channels develop first near the pulp surface, or if transverse cracks form in the pulp, the critical dilution is higher than that of the test. The exact critical dilution is indicated by the simultaneous formation of channels in all parts of the settling column, the channels in the upper part of the column being usually replaced later by the extensions of the channels from the bottom of the vessel.

For determining settling rates in the free-settling zone, cylinders are filled to a depth of 1 ft. with various dilutions of pulp, one sample being at exactly critical dilution, the remaining samples representing an ascending scale of dilutions above the critical point. The $L:S$ ratio of each sample is determined by test or by calculation. Each sample is violently agitated and the settling tests started.[14] From the first 0.05 ft. settled

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[13] Calcium sulphate, derived from the action of lime hydrate on iron sulphate, is an active flocculating agent.

[14] In determining minimum settling equipment, it is essential for temperature and electrolyte in the tests to represent minimum conditions in the plant.

in the tests, are calculated the settling rates corresponding to the feed horizons of continuous-settling tanks. These may be termed "the settling rates of feed." From the second 0.05 ft. settled in the tests (of lower dilutions) are calculated the settling rates corresponding to the various free-settling dilutions in the depths of continuous-settling tanks. These may be termed the "free-settling-rates."

Settling rates in the thickening zone are determined in a single test. For this purpose a cylindrical graduate, of 1 liter, or greater, capacity, is marked at the following depths:

*Marks on Testing Cylinder*[15]

| Depths in Feet | Depths in Feet |
|---|---|
| 1.160 | 0 636 |
| 1.050 | 0 576 |
| 0.950 | 0.521 |
| 0.860 | 0 471 |
| 0.778 | 0.427 |
| 0.704 | 0.386 |

Each depth is $^{95}\!/_{105}$ of the previous one. The distance between any two consecutive depths is one-tenth the average of the two depths. Hence the distance settled between any two consecutive marks represents $1/10$ ft., per foot depth of pulp. This principle is of great assistance in calculating settling rates in the thickening zone.

The specially marked cylinder is filled to the 1.05-ft. mark with pulp at exactly critical dilution. Solution is added until the combined pulp and solution reaches the 1.16-ft. mark. The mixture is then agitated and the test started. Due to the addition of solution, the test begins in the free-settling zone, and flocs are at least partially formed before the critical dilution is reached. This somewhat parallels usual plant practice and gives more representative results than are obtained when the test is started at exactly critical dilution. The time is first noted as the pulp surface passes the 1.05-ft. mark, and is thereafter noted for each subsequent mark until settling ceases. This test indicates the lowest dilution that can be obtained in the tank discharge, and furnishes the basis for calculating the depth of tank required to produce that dilution, or any other dilution in the thickening zone. The settling rates in the thickening zone must represent the average rate of subsidence of pulp surface, per foot depth of pulp, in settling from the critical dilution to the various possible dilutions of discharge.

The average settling rate, corresponding to any dilution of discharge, may be quickly calculated by considering each space as $1/10$ ft., and by dividing the aggregate space settled (so considered) by the time required for the pulp surface to settle from critical dilution to the dilution con-

[15] In case there is a contraction in the bottom of the cylinder, spaces are proportioned to volume rather than depth.

sidered. The method for tabulating observations and calculating settling rates is illustrated in Table 1.

The dilutions of pulp, corresponding to the various marks on the thickening cylinder, may be successively calculated from the critical dilution by the following formula:

$$D = 0.905\,C - \frac{0.095}{G} \qquad \text{(Formula 2)}$$

$D$ and $C$ are the $L:S$ ratios corresponding to any two consecutive depths.

$G$ is the specific gravity of dry pulp.

This method for calculating dilutions may be extended to the free-settling tests by similarly marking all cylinders, the range of free-settling dilutions being secured by filling the various vessels to the different depths with pulp at critical dilution and adding solution to each vessel until the 1.05-ft. mark is reached.

*Determination of Area and Depth of Continuous-Settling Tanks*

The determination of the settling area, required for any definite settling operation, is based on the principle that the elimination of water in the free-settling zone is a function of settling area and is independent of pulp depth. Expressed as a formula, this law is:

$$A = \frac{1.34\,(F - D)}{S} \qquad \text{(Formula 3)}$$

Where

$A$ = Area in square feet per ton of dry pulp settled daily.

$F$ = $L:S$ ratio of the governing dilution.

$D$ = $L:S$ ratio of the dilution of discharge.

$S$ = Settling rate of governing dilution (as determined by laboratory test at free-settling dilutions).

The numerical constant (1.34) is the factor introduced by reducing the quantity of water eliminated, from cubic feet per hour to tons per day.

The derivation of this formula is explained more fully by Coe and Clevenger, and also in a former article on the settling of Tigre slime.[16] The idea of a governing dilution (introduced by Coe and Clevenger) generalizes the formula, rendering it applicable, should area be governed by some free-settling rate in the depths of a settling tank.

As an example of the application of the above formula, the area of tank required to settle Tigre pulp from a dilution of 14 : 1 to a dilution of 2.3 : 1, may be calculated from the data presented in Table 1. The settling rate of the feed, determined by interpolation in the table, is 0.78 ft. per hour. Substituting in Formula 3:

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[16] *Engineering and Mining Journal* (Oct. 5, 1912), **44**, 1643.

$$A = \frac{1.34\,(14 - 2.3)}{0.78} = \begin{array}{l}\text{20 sq. ft. per dry ton settled daily} \\ \text{(based on the settling rate of the feed).}\end{array}$$

When the same formula is applied to the various free-settling dilutions between 14 : 1 and 5.1 : 1 (critical dilution), the largest area required is 16 sq. ft.—corresponding to the dilution of 13.1 : 1. This is considerably less than the area determined by the feed dilution. Hence in this case, the settling rate of the feed is the governing factor, the required area being 20 sq. ft. per ton of dry pulp settled daily.

The determination of the depth required in continuous-settling tanks depends on the law that the elimination of water in the thickening zone is a function of both area and depth. This law is expressed in the following formula:

$$\text{Depth} = \frac{1.34\,(C - D)}{A \times T} \qquad \text{(Formula 4)}[17]$$

Where

$C$ and $D$ are respectively the $L:S$ ratios of critical dilution and of discharge.

$A$ is the settling area (as previously determined by Formula 3).

$T$ is the average settling rate, per foot depth of pulp, in settling from dilution $C$ to dilution $D$ (determined in laboratory tests, previously described).

In the problem to be solved:

$C = 5.1$; $D = 2.3$; $A = 2.0$; and $T = 0.03$ (from Table 1).

Substituting in Formula 4:

$$\text{Depth} = \frac{1.34\,(5.1 - 2.3)}{20 \times 0.03} = 6.3 \text{ ft.}$$

From the above calculations, it appears that in order to settle Tigre pulp from a feed dilution of 14 : 1 to a dilution of discharge of 2.3 : 1 (the lime being 0.3 lb. per ton and the temperature 75° F.) it is necessary to provide a minimum area of 20 sq. ft. per ton of dry pulp settled daily, and a minimum effective depth of 6.3 ft. This checks closely with actual requirements in the plant.

The method outlined above is applicable for any other dilutions of feed and discharge within the limits of the tests.

Problems, involving the determination of dilutions of feed or discharge when area and depth are known, may be solved by transposing or combining Formulas 3 and 4.

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[17] The formula may be mathematically derived by applying Formula 3 to the horizon of critical dilution.

This method for calculating depth, and the method suggested by Coe and Clevenger, were developed quite independently. Each is based on a different viewpoint of the same law. Hence the results from both methods are identical.

## Graphical Method for Solving Settling Problems

In designing settling equipment it is often necessary to consider numerous combinations of interdependent settling operations. The algebraic solution of such problems is cumbersome, involving a great bulk of calculations for each combination considered. A graphical method, designed to meet the needs at Tigre, might be suggested for use in such cases.

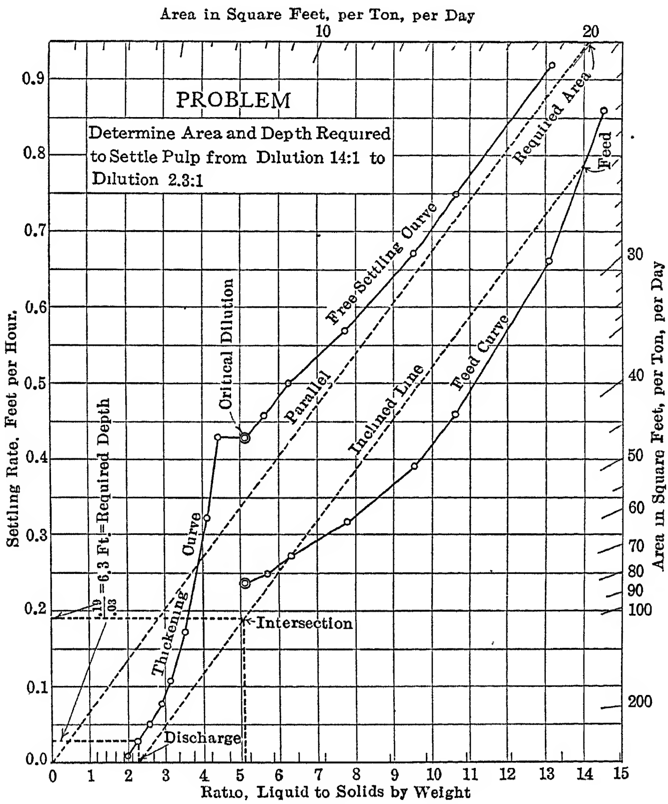


Fig. 1.

For the graphical method, settling tests are performed as previously described, the specially marked cylinders being used in all tests. The settling rates, corresponding to the various dilutions, are plotted on a printed chart (see Fig. 1). The vertical axis of the chart represents settling rates. The horizontal axis is calibrated for both $L:S$ ratios and divisions marked on the cylinders (the short marks above the axis representing the marks on the cylinders). With the $L:S$ ratio of the critical dilution known, lower or higher dilutions may be determined by

counting divisions to left or right, thus obviating the necessity for calculating the ratios of liquid to solids, corresponding to the various dilutions. The radial calibrations at the upper and right-hand margins of the chart are for the purpose of graphically solving problems of area. The problem solved algebraically above is solved graphically in Fig. 1.

Following is the standard procedure for determining area and depth, when dilutions of feed and discharge are known:

1. From the point on the horizontal axis representing dilution of discharge, draw an inclined line to the point on the curve representing settling rate of feed. In case the inclined line intersects the free-settling curve, it is rotated downward until tangent to the lowest depression on the curve. In this case, the point of tangency represents the dilution which governs area.[18] If the inclined line does not intersect the free-settling curve, area is governed by the settling rate of feed.

2. Parallel to the inclined line, draw a line from the zero point of the axis to the margin of the chart and read the area required for each ton of dry pulp, settled daily.

3. Observe the settling rate indicated by the intersection of the inclined line with the ordinate through the critical dilution. Divide this settling rate by the settling rate shown on the curve, corresponding to the dilution of discharge. The quotient is depth required.

Problems involving the dilution of feed or discharge, when depth and area of tank are known, may be solved by reversing this procedure.

The fundamental principles underlying the graphical calculations are identical with those upon which the algebraic determinations are based. Results are the same when calculated by either method. The advantages of the graphical method are greater simplicity, greater rapidity, and the possibility of solving a larger number of problems from a single set of observations. The graphical method is of most service in solving complex problems, in which various settling operations are interdependent.

## Scope of Laboratory Tests

The laboratory methods for ascertaining the required dimensions of continuous-settling tanks are by no means perfect. Small errors are undoubtedly introduced by the empirical manner of experimentally determining the effects of the rate of flocculation on settling. Other errors possibly enter on account of the failure to determine the extent to which the pulp in the upper part of the tank is diluted by water rising from below. These errors are not serious. Quite possibly they may be altogether eliminated by future investigations. In the meantime, the methods already developed may be employed with the assurance of obtaining reasonably accurate results.

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[18] The graphical method is based on the principle that the cotangent of the angle, between the horizontal axis and the inclined line, is equal to the factor $\frac{F - D}{S}$ of Formula 3.

The particular advantage of laboratory determinations lies in the rapidity with which they may be performed. A complete series of tests, covering all necessary dilutions of feed and discharge, usually requires 3 hr. of constant attention, followed by 6 hr. of intermittent attention, and perhaps a couple of observations the next day. In 3 or 4 days it is often possible to cover, in a general way, all conditions of dilution, electrolyte, and temperature, that might occur in the plant.

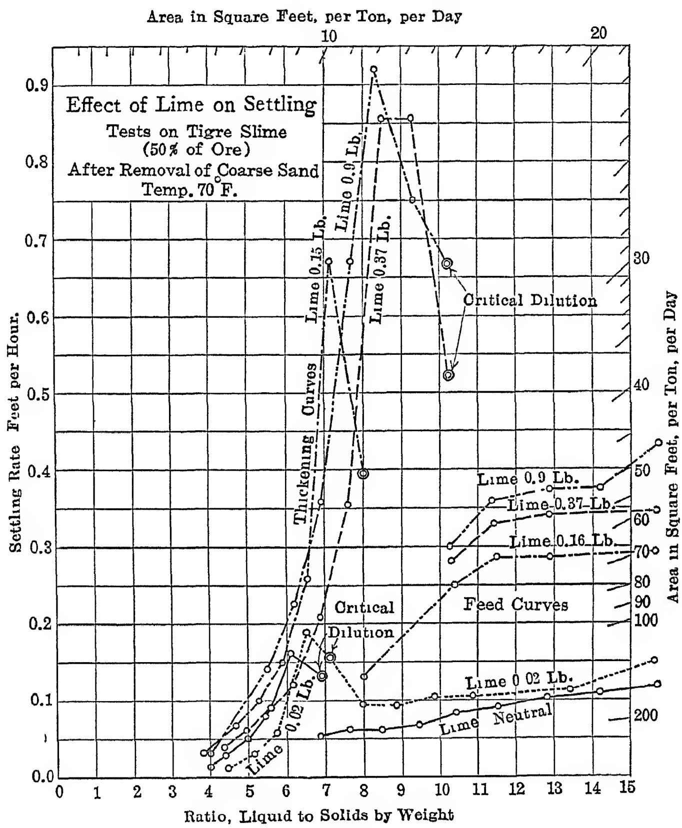


FIG. 2.

Laboratory tests are most useful for preliminary investigations prior to mill construction. They furnish means for roughly estimating the settling equipment necessary for various requirements and under varied conditions in plant practice. These data, combined with similar information regarding dependent operations, enable the mill designer to decide intelligently upon the conditions to be maintained in the plant, and the duty to be performed by the settling equipment. It is as an aid in deciding upon the flow sheet of a contemplated plant, that laboratory settling tests are particularly useful. The approximate determination of the

settling equipment necessary to fulfil the requirements imposed by the flow sheet, is a secondary advantage.

## Unit Tests

Before proceeding with extensive installations, it is usually advisable to check, by unit tests, the laboratory determinations of the required settling equipment. Unit tests should preferably be performed in tanks of full height, though the area may be considerably less than that of the

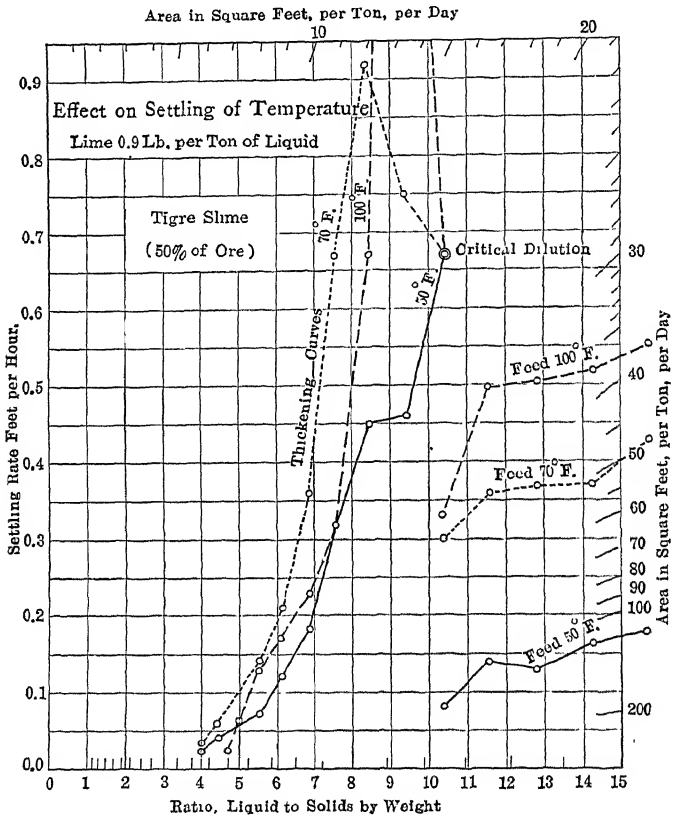


Fig. 3.

tanks to be installed. Test tanks, for determining the size of small installations, may be constructed of 12- or 16-in. pipe, suitable overflow, feed column, cone bottom, and continuous discharge being provided. For determining extensive installations, the tests should preferably be conducted in standard settling tanks. Observations in unit tests may be started as soon as the required dilution is secured in the discharge, and may be continued over periods ranging from a day to a month, the dura-

tion of the tests depending upon the accuracy desired and the importance of the equipment to be installed. It is important that the required conditions of dilution, electrolyte, temperature, and pulp depth, be maintained throughout the tests; and that the capacity, under these conditions, be accurately ascertained. In determining settling equipment from unit tests, capacity is regarded as proportional to settling area.

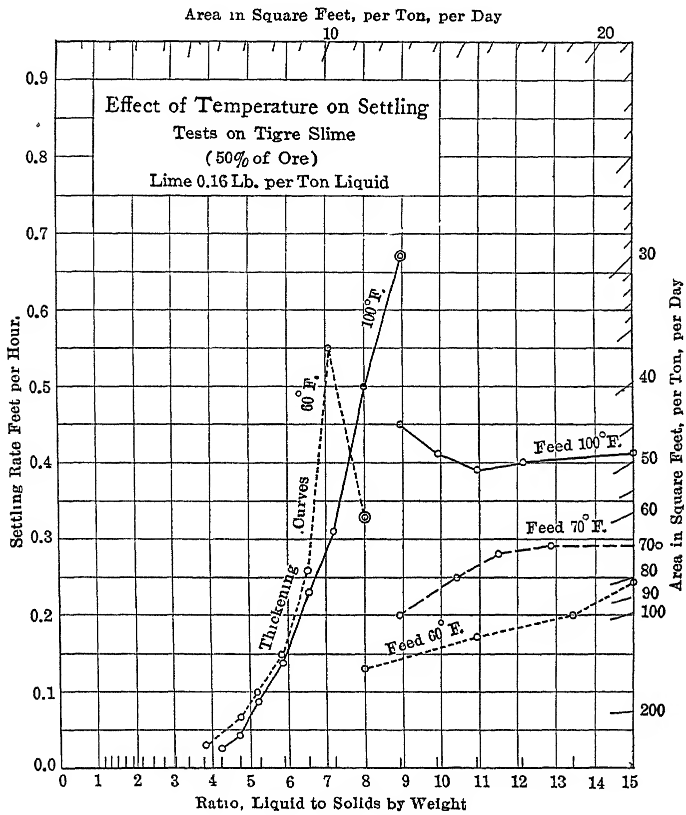


Fig. 4.

## Various Factors Affecting Settling

Repeated mention is made in the foregoing discussion of the importance, in settling operations, of dilution and temperature. The effects of variations in these conditions, upon sundry phases of settling at Tigre, are graphically illustrated in Fig. 2, 3 and 4. The data, presented in these figures, may be briefly summarized as follows:

### *Effects of Dilution*

In the free-settling zone, settling rates are greatest at high dilutions and lowest at low dilutions.

In the thickening zone, settling rates are greatly increased at dilutions just below critical dilution, reaching a maximum when the dilution is four-fifths that of critical dilution. Below this dilution, settling rates rapidly decrease as the dilution is reduced.

### *Effects of Lime*

In the free-settling zone, settling rates of feed are trebled by increasing lime from 0 to 0.2 lb. per ton of solution, and are quadrupled by increasing lime from 0 to 0.7 lb. Further addition of lime above 0.7 lb. produces little advantage.

The critical dilution is doubled by increasing lime from 0 to 0.4 lb. Further addition of lime produces little effect on critical dilution.

At the higher dilutions in the thickening zone, lime up to 0.4 lb. greatly increases settling rate.

At the lower dilutions in the thickening zone, variations in lime have little effect on settling rate, except in very shallow columns, or in the absence of classified sand.

### *Effects of Temperature*

In the free-settling zone, an increase in temperature from 40° to 80° F. trebles settling rates.

Critical dilution increases slightly as the temperature is raised.

At the higher dilutions in the thickening zone, increase in temperature from 40° to 100° doubles settling rates.

At the lower dilutions in the thickening zone, changes in temperature have little effect on settling rates.

## General Conclusions

High dilution, low lime and low temperatures predicate extensive settling area and tanks of shallow depth.

Low dilutions, high lime, and high temperatures predicate less area and comparatively deep tanks.

Adequate area is the first consideration in settling. Any advantage from increased depth can also be obtained from increased area. On the other hand, when area is inadequate, tank depth has no effect on results of settling.

## Settling Problems at El Tigre

Recent changes in treatment at El Tigre necessitated the reduction of lime in the mill water, from 1.5 lb. per ton to 0.02 lb. Before the changes were made, it was realized that this reduction in electrolyte

would greatly decrease the already low settling rate of Tigre pulp. Hence all conditions affecting settling were carefully reëxamined.

The ore at Tigre is crushed in stamp batteries to pass a 2-mesh screen. It is then ground to 20-mesh in a Hardinge mill. After concentration, it is further reduced, in tube mills, until 85 per cent. will pass a 200-mesh screen. The all-slime product is then settled to the $L:S$ ratio of 5 : 1.

Tigre ore contains 25 per cent. natural slime. In the presence of this proportion of slime, classification of fine sand in the tube-mill circuit can be accomplished only at dilutions above 15 : 1.

To settle Tigre pulp, in the absence of strong electrolytes, requires extremely large settling area. This is especially true during the cold winter months. Preliminary laboratory tests indicated that in order to settle average Tigre ore from a dilution of 15 : 1 to a dilution of 5 : 1, the lime being 0.02 lb. and the temperature 45° F., it would be necessary to provide 110 sq. ft. of settling area per ton of ore settled daily.[19] On this basis, 61 settling tanks, each 24 ft. in diameter, would be required to handle the Tigre tonnage of 250 tons per day.

The problem was solved, not by supplying this excessive equipment, but by modifying conditions elsewhere in the plant.

The dilution at which classification of sand can be accomplished in mechanical classifiers depends upon the coarseness of the sand to be separated and the proportion of natural slime in the pulp.[20] To separate 20-mesh sand, dilutions between 4 : 1 and 1 : 1 are usually required. The classification of finer sand necessitates higher dilutions. With 25 per cent. natural slime in the pulp, 200-mesh sand can be separated only at dilutions above 15 : 1. When the pulp contains 8 per cent. natural slime, satisfactory classification of 200-mesh sand can be effected at the dilution of 6 : 1. In the presence of 1 per cent. natural slime, classification is satisfactory at the dilution of 1 : 1. It thus appears that classification is a function of the dilution of the natural slime and is practically independent of the proportion of sand occurring in the pulp.

Settling rates, also, vary inversely with the percentage of natural slime in the pulp. It follows, therefore, that in order to obtain a thick, "all-slime" product from an ore containing an excessive amount of natural slime, it is important to separate the natural slime and sand, while the slime is undiluted and the sand is still coarse. The coarse sand can then be diluted sufficiently to insure good classification in the fine-grinding department of the plant, without necessitating extensive equipment for dewatering the product.

These objects are accomplished at El Tigre by modifications in the

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[19] This estimate has been subsequently verified by plant operations.

[20] Classification depends also on temperature and electrolyte, being most satisfactory when temperature and strength of lime are low. Any condition, tending to retard or prevent flocculation, aids classification.

stamp mill. The batteries are divided into two units, a mechanically operated classifier receiving the discharge from each unit. The sand from both classifiers is sluiced to the Hardinge mill with fresh water. Attrition in the Hardinge mill produces a certain amount of natural slime—from the disintegration of clay balls, etc. This is washed from the sand in hydraulic classifiers. The sand from the hydraulic classifiers is concentrated, and flows by gravity to the tube-mill room.

The slime overflow from the hydraulic classifiers is elevated and used as feed water in the first unit of batteries. The slime overflow from the classifier below the first unit of batteries is elevated and employed as feed water in the second unit. Thus the battery feed water is used in three operations, acquiring a load of slime in each operation. The final slime product, overflowing from the classifier below the second unit of batteries, has a dilution of 6 : 1 and requires no further thickening.

On account of the low dilution of the final slime product, much fine sand is carried over with the slime overflow of the second classifier. This is separated on Deister slime tables, which are equipped with deep riffles for the purpose. The shallow flow of slime across these tables and the sharp movements imparted by the mechanism prevent flocculation and effect a good separation of sand from the thick pulp. The slime tailing from the Deister tables tests 87 per cent. through 200-mesh, which is satisfactory. The sand product is concentrated on a recleaning table, equipped with standard riffles. Improved concentration is secured on the Deister tables by this roughing and recleaning process. The final sand tailing from the recleaning table joins the sand from the upper part of the mill and flows by gravity to the tube mills.

The low dilution in the battery discharge necessarily reduces tonnage through the batteries. This effect is counteracted by employing coarser screens and by using the same feed water successively in the two sets of batteries. An average daily tonnage of 12 tons per 1,200-lb. stamp is maintained.

Low efficiency in the batteries is offset by improved efficiency in the Hardinge and tube mills.

The cost of the successive elevations of slime in the stamp mill is offset by reduced cost of pumping from the bottom of the mill.

The sand entering the tube-mill room still contains some natural slime. More slime is liberated in the tube mills by the disintegration of partially kaolinized feldspar. The final overflow from the tube-mill classifiers contains 7 per cent. natural slime, and 93 per cent. fine sand. This product tests 85 per cent. through 200-mesh. The dilution is 7:1. It is settled to a dilution of 4 : 1 in 12 shallow settling tanks, each 24 ft. in diameter.

The combined dilution of final sand and slime products is 5:1, which is the dilution required.

Lime, 0.3 lb.
Temp., 73° F.

TABLE 1.—*Laboratory Tests on Tigre Pulp*

| Test No. | L:S Beginning of Test | L:S End of Test | D Depth of Clear Liquid Formed per Foot of Depth, Feet | T Duration of Test, Minutes | Settling Rate $\frac{60D}{T}$, Feet per Hour |
|---|---|---|---|---|---|
| Settling rates of feed: | | | | | |
| 1 | 14.5 | ....... | 0 05 | 3 50 | 0.86 |
| 2 | 13 1 | ....... | 0.05 | 4.50 | 0.66 |
| 4 | 10 7 | .... | 0.05 | 6.50 | 0.46 |
| 5 | 9.6 | ... | 0.05 | 7.75 | 0.39 |
| 7 | 7.8 | .... | 0.05 | 9.50 | 0.32 |
| 9 | 6 3 | ....... | 0.05 | 11.00 | 0 27 |
| 10 | 5 7 | .... | 0 05 | 12 00 | 0.25 |
| 11 | 5.1* | ..... | 0.05 | 12.50 | 0.24 |
| Rates of free settling: | | | | | |
| 1 | 14 5 | ....... | 0.05 | 2 50 | 1 20 |
| 2 | 13.1 | ....... | 0.05 | 3 25 | 0 92 |
| 4 | 10.7 | .... | 0.05 | 4.00 | 0 75 |
| 5 | 9.6 | ........ | 0.05 | 4.50 | 0.67 |
| 7 | 7.8 | ....... | 0.05 | 5.25 | 0 57 |
| 9 | 6.3 | ..... | 0.05 | 6.00 | 0.50 |
| 10 | 5.7 | ..... | 0.05 | 6.50 | 0.46 |
| 11 | 5.1* | .. | 0 05 | 7.00 | 0.43 |
| Thickening test: | | | | | |
| 12 | 5.1* | 4.6 | 0.10 | 14.00 | 0.43 |
| 13 | 5.1 | 4.1 | 0.20 | 37.00 | 0.32 |
| 14 | 5.1 | 3.7 | 0.30 | 108.00 | 0.17 |
| 15 | 5.1 | 3.3 | 0.40 | 223.00 | 0.11 |
| 16 | 5.1 | 2.8 | 0.50 | 383.00 | 0 08 |
| 17 | 5.1 | 2.5 | 0.60 | 720.00 | 0.05 |
| 18 | 5.1 | 2.3 | 0.70 | 1,380.00 | 0 03 |
| 19 | 5 1 | 2 0 | 0.80 | 4,520.00 | 0 01 |

* Critical dilution.

Thus the problem was solved by the installation of one-fifth the settling equipment estimated as necessary, and without sacrificing any element of efficiency in mill operations.

In the above discussion, an effort has been made to submit, as briefly as possible, any results of investigation at El Tigre which might have application elsewhere. Since tests have been limited to one class of ore, many of the observations can have only local importance, or at the most,

TABLE 2.—*Actual Settling Rates in Tank Compared with Settling Rates Determined by Test*

Tests on Tigre pulp (50 per cent. fine sand, 50 per cent. slime).
Observations in plant, over a period of 3 months.
Observations are averaged for days having similar temperature, dilution of feed, and amount of lime in solution.

| No. of Days Averaged | Lime in Solution, Pounds per Ton | Temperature °F. | Square Feet Settling Area per Ton per Day | $R_1$ L:S Feed | $R_2$ L:S Discharge | Settling Rate, Feet per Hour $\frac{1.34(R_1 - R_2)}{A}$ | Settling Rate from Laboratory Test |
|---|---|---|---|---|---|---|---|
| 1 | 0.15 | 45 | 14.0 | 4.75 | 3.70 | 0.10 | 0.11 |
| 7 | 0.15 | 55 | 14.9 | 4.75 | 2.75 | 0.18 | 0.15 |
| 3 | 0.15 | 55 | 15.5 | 5.48 | 3.30 | 0.19 | 0.17 |
| 1 | 0.15 | 55 | 23.0 | 7.00 | 3.87 | 0.18 | 0.21 |
| 2 | 0.15 | 65 | 9.5 | 3.44 | 3.12 | 0.05 | 0.06 |
| 2 | 0.15 | 65 | 12.4 | 4.46 | 2.80 | 0.18 | 0.17 |
| 1 | 0.15 | 65 | 13.6 | 5.16 | 3.16 | 0.20 | 0.19 |
| 4 | 0.25 | 45 | 9.5 | 4.23 | 3.10 | 0.16 | 0.11 |
| 6 | 0.25 | 55 | 12.2 | 4.76 | 3.10 | 0.18 | 0.15 |
| 3 | 0.25 | 55 | 12.7 | 5.23 | 3.46 | 0.19 | 0.16 |
| 2 | 0.25 | 55 | 12.4 | 7.05 | 4.21 | 0.31 | 0.22 |
| 8 | 0.25 | 65 | 11.2 | 4.26 | 2.88 | 0.16 | 0.16 |
| 10 | 0.25 | 65 | 11.1 | 4.73 | 2.83 | 0.23 | 0.20 |
| 5 | 0.25 | 65 | 12.2 | 5.41 | 3.25 | 0.24 | 0.24 |
| 1 | 0.25 | 75 | 9.1 | 3.85 | 3.17 | 0.10 | 0.13 |
| 3 | 0.25 | 75 | 10.8 | 4.64 | 2.69 | 0.24 | 0.22 |
| 6 | 0.25 | 75 | 12.0 | 5.52 | 3.69 | 0.21 | 0.25 |
| 3 | 0.25 | 75 | 13.0 | 6.68 | 4.30 | 0.24 | 0.28 |
| 1 | 0.35 | 55 | 9.2 | 3.48 | 3.08 | 0.06 | 0.06 |
| 1 | 0.35 | 55 | 22.0 | 6.40 | 3.00 | 0.21 | 0.19 |
| 1 | 0.35 | 65 | 14.2 | 3.64 | 3.16 | 0.05 | 0.08 |
| 8 | 0.35 | 65 | 11.4 | 4.56 | 2.81 | 0.20 | 0.19 |
| 5 | 0.35 | 65 | 11.5 | 5.29 | 2.74 | 0.30 | 0.26 |
| 1 | 0.45 | 65 | 10.8 | 3.86 | 2.84 | 0.13 | 0.12 |
| 5 | 0.45 | 65 | 11.3 | 4.61 | 2.72 | 0.22 | 0.22 |
| Average. | | | | | | 0.18 | 0.17 |

NOTE.—Critical dilutions: 4.0 : 1 at 0.15 lb. lime.
4.7 : 1 at 0.25 lb. lime.
5.1 : 1 at 0.35 lb. lime.
5.2 : 1 at 0.45 lb. lime.

serve to indicate what may be expected under like conditions elsewhere. In other instances, the investigations at El Tigre and the more comprehensive investigations of Coe and Clevenger, are mutually confirmatory,

TABLE 3.—*Thickening Test on Tigre Slime (47 Per Cent. of Ore)*

(All possible sand removed by previous classification)

Temperature 75° Saturated lime solution

Tests begun below critical dilution and continued until settling ceased.

Tests performed in glass cylinders (1/10 ft. diam.).

| Test No. | L : S Ratio | | Average Settling Rates: Feet per Hour for Different Column Depths | | | | |
|---|---|---|---|---|---|---|---|
| | Beginning of Observation | End of Observation | 3 in. | 6 in. | 12 in. | 24 in. | 48 in. |
| 1 | 7.7 | 6 9 | 0 060 | 0.110 | 0.180 | 0.340 | 0.340 |
| 2 | 6.9 | 6.3 | 0.040 | 0.060 | 0.080 | 0 300 | 0.600 |
| 3 | 6.3 | 5.6 | 0 030 | 0.040 | 0.070 | 0.140 | 0.680 |
| 4 | 5.6 | 5 0 | 0.021 | 0.037 | 0.047 | 0.066 | 0.424 |
| 5 | 5.0 | 4.5 | 0.015 | 0.028 | 0.033 | 0.056 | 0.088 |
| 6 | 4.5 | 4.1 | ....... | 0.023 | 0.022 | 0.022 | 0.020 |
| 7 | 4.1 | 3.6 | ....... | ....... | 0 010 | 0.009 | 0.009 |
| 8 | 3.6 | 3.2 | ....... | ....... | ....... | 0.004 | 0 005 |

No classification of sand observed.

No channels in 3-in. column.

Short channels in upper part of 6-in.

Strong channels in 12-in., 24-in. and 48-in. columns, but these closed during last part of test.

Only short channels (less than 2 in. long) occurred near the surface in the observations below the heavy line.

Settling rates below heavy line are independent of pulp depth.

No arching of the slime occurred and no cracks or pockets of clear liquid could be detected at the end of the tests.

making it possible to formulate, and employ with confidence, many general principles of settling.

Although all phases of settling have by no means been investigated, sufficient work has been done to render possible the formulation of certain standard methods for the solution of settling problems. Undoubtedly these methods will be simplified and rendered more accurate by future investigations.

## DISCUSSION

H. S. COE, St. Louis, Mo. (written discussion*).—I feel much gratified that Mr. Mishler has confirmed those principles of slime settlement expressed by Mr. Clevenger and myself. I wish to state that when we wrote the article referred to by Mr. Mishler we had not seen his earlier article in which he gave a formula to be applied to the settling rate of feed pulp for determining thickener area required at El Tigre. It has

* Received Oct. 8, 1917.

been my experience that in cyanide work the settling area required is determined by the settling rate of the feed pulp for perhaps 20 per cent. of all of the ores to be encountered. For the other ores, the area required is greater than that indicated by the settling rate of the feed and corresponds to the settlement of a somewhat thicker free-settling pulp. Mr. Mishler, in his paper, has thoroughly outlined a set of tests for determining thickener capacity, which, if carefully carried out, should be entirely reliable, provided allowance is made for changes in the mill feed from time to time.

Two features of his paper, I believe, need comment. In speaking of the time required for slime to flocculate after being fed, Mr. Mishler quite properly states that the actual settling rate for the feed pulp cannot be that of completely flocculated pulp because of this time lost in flocculation. It seems to me, however, that this feature cannot be of very great importance, for the following reason: The upper zones of pulp in a thickener may be, and often are, as thin as the feed pulp. If, therefore, the feed be introduced at sufficient depth below the pulp surface to prevent agitation at the surface, there will be ample time for it to become flocculated before it becomes the surface layer, and it will therefore cause no appreciable retardation in the settlement of the surface of the pulp.

Mr. Mishler speaks of the stage of settlement in which, after the pores and water channels collapse, further settlement depends again directly on the area of the container and not on the volume. I have often observed this to be true, and where settlement is required to a density of discharge thicker than that at which the channels close, increased depth is not advantageous for a tank, but rather increased area, as has been demonstrated in settling Anaconda flotation concentrates. It seems highly probable that in the final stage a stirring of the pulp, such as is accomplished by the thickener mechanism, will increase the rate of settlement; and since this final settlement is extremely slow, sometimes amounting to almost nothing when not stirred, the average settling rate in the final thickening stage may be so increased, by stirring, as both to diminish the depth of tank required and to increase the thickness of the final discharge.

Mr. Bloomfield, in his work with shallow Dorr trays at the Golden Cycle mill, reports results which strongly indicate this to be the fact for Golden Cycle ore. Stirring would not increase the capacity of the thickener beyond the natural capacity for discharging pulp of the consistency at which the channels have just closed; on the contrary, the capacity should be slightly lower, due to the fact that the $D$ in Mr. Mishler's formula No. 3 becomes less and $A$, or required area, consequently becomes greater.

Another phase of slime settlement which has not been discussed

should be spoken of. I refer to the clarification of water or the settling of very thin pulps too dilute to flocculate in the ordinary way. I have found that a 10-ft. depth of turbid water will become clear in approximately the same time as a 1-ft. depth.

In the cases observed, the minute particles in suspension are drawn together in an accidental manner to form floccules which fall to the bottom. The water thus clears by degrees and all of the particles will have flocculated and settled from a large volume in about the same time as would be required for a small volume. In clarifying water for domestic use, the basins are designed to contain water for what is designated as "the detention period," or time required to clear the water.

Whenever very clear overflows are required from thickeners, the tank should be constructed to retain the solution above the pulp level for the required detention period. That is, the tanks should be sufficiently deep to allow the requisite zone of clarifying liquid above the pulp.

# Tests on the Hardinge Conical Mill

BY ARTHUR F. TAGGART,* NEW HAVEN, CONN.

(St. Louis Meeting, October, 1917)

## Introduction

THE major portion of the work described in this paper was performed by R. W. Young,† a graduate student in the department of Mining and Metallurgy, Sheffield Scientific School, Yale University, working under a coöperative agreement between the Hardinge Conical Mill Co., the Sheffield Scientific School, and himself.

Since this coöperative scheme is at present in effect in the case of a considerable number of other students in the department and since it is the hope of the school that the privilege thus extended may be utilized even more freely in the future by mining and manufacturing companies, it may not be amiss at this point to give a summary of the general plan. Briefly it is as follows:

A graduate student, whose undergraduate work in this or other universities shows promise of ability to handle research work, is chosen by conference between the company and instructor involved. The aim of the company in the agreement then entered into is to obtain the solution of one or more of the technical problems with which it may be confronted, or, at the end of 1 or 2 years, to obtain as an employee a man especially trained in its work. As a means to accomplish one or both of these ends, the company furnishes the machine, apparatus, or material to be tested and pays the student during his graduate work a small salary, usually just sufficient to cover his living expenses, tuition and fees. The aim of the student is special training along a line in which he is particularly interested, the attainment of his advanced degree, and the chance to show to his future employer ability to handle such problems as may be presented to him. In return for the financial aid which he receives he agrees to devote at least half of his working time to the special problem submitted by his company. The other half is devoted to study of the collateral subjects required by the department for the granting of the degree which the student seeks. The student further agrees to enter the employ of the company in question at a wage not greater than that paid in like positions to recent graduates not specially trained and to remain with his employer at such wage for at east 1 year. If the student is to obtain a degree, the special work forming the basis of his investigation must be such as will involve real research and not mere routine manipulation. The subject is chosen by conference between the three parties to the agreement. The work is carried on under the direct supervision of the instructor involved. The school furnishes the general laboratory and library equipment essential to the pursuit of any extended investigation. In

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† Deceased, June, 1916.

return it is expected that the results of the investigation shall be, in part at least, available for publication, if they are deemed of interest to the profession.

## Mill Used

In the laboratory work described in the following pages, a 4½-ft. Hardinge mill with three removable cylindrical sections, 16 in. each in length, was used. Fig. 1 shows the mill with three cylindrical rings in place. This combination allows a mill 4½ ft. by 0 in., 4½ ft. by 16 in., 4½ ft. by 32 in., or 4½ ft. by 48 in., as desired. The conical and cylindrical sections were built of cast iron, 1¼ in. thick and were lined with chrome-

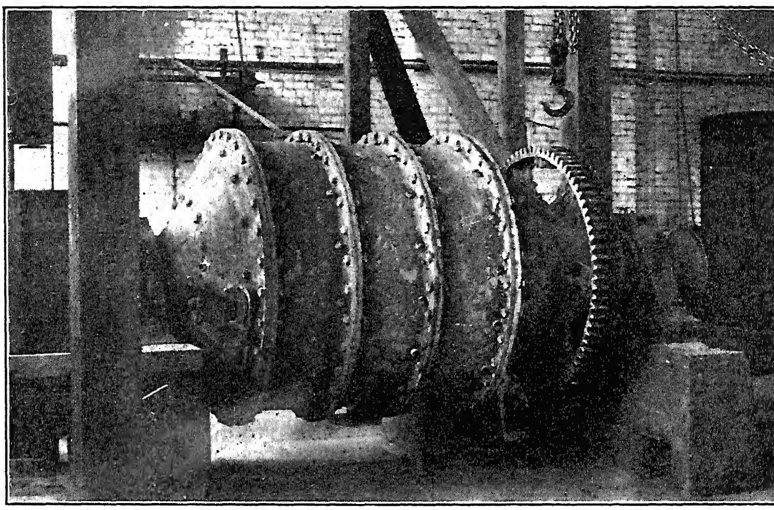


Fig. 1.—Hardinge mill used in grinding tests.

steel lifting bars 2½ in. high, 3 in. wide and 16 in. long, set on 11-in. centers. The head bearing was adjustable in height, thus allowing the mill to be tilted any desired amount.

## Material

A majority of the tests were made on quartzite and trap. The quartzite contained an appreciable amount of white mica in flakes 1 to 2 mm. (0.04 to 0.08 in.) diameter, which made it rather easy to crush in the coarser sizes but difficult to grind when the finer sizes were reached. The trap was a variety of diabase quarried locally for road metal. The other materials tested (see tests 230 to 236) were of a special nature and will be described more particularly in connection with the record of the work done upon them.

## Method of Feeding

The material to be ground in any given test was weighed up and divided into lots, each lot being sufficient to furnish the feed for the mill for 5 min. In general, this lot was divided by eye into five portions, so that a portion could be dumped into the feed box every minute. This procedure, aided by the low capacity of the scoop feeder on coarse material, assured a practically uniform feed rate. When, as was the case with finer materials, the scoop tended to take up the material meant for a 1 min. portion in two or three revolutions, the method of feeding was so changed as to take away from the scoop the burden of regulating the feed for even such a short interval as 1 min. The importance of this insistence on regular feed will be seen in Fig. 10, which presents a comparison of the feed and discharge rates of the mill, dry crushing. In wet feeding the same methods of introducing the rock were followed. The water was introduced into the feed box from a calibrated orifice at the proper rate to give the desired moisture percentage and the result was checked by moisture samples of the discharge.

## Sampling

Feed samples were taken, in every case, by the method of alternate shovels. Large samples were cut to insure accuracy. Samples of the product consisted of the whole discharge stream caught for varying intervals according to the feed rate. The interval for wet samples was rarely less than 1 min. For dry samples the interval was never less than 1 min. and in all cases where the feed rate was less than 1 ton per hour the sample of the product consisted of the whole discharge for an interval of 5 min.

## Screen Testing

Screen tests on feed samples were made in duplicate. The accuracy of the sampling was accepted as sufficient when cumulative graphs of the tests were closely coincident. Product samples were passed over the 6.680-mm. (0.26-in.) sieve. The total oversize on this sieve was then run through the coarser series. The undersize of the 6.680-mm. sieve was riffled down to not less than 200 gm. and then run through the remainder of the Tyler Standard Sieve Scale series of screens (1.414 ratio). The amounts of the aliquot parts of the whole sample remaining on these fine sieves and passing the last (0.074 mm.) were then calculated back into terms of the whole sample and the percentages given in Table 2 were calculated from the figures thus determined. Duplicate samples of the riffled undersize were run occasionally in order that frequent screen tests might not breed carelessness. In no case was the difference between duplicates greater than that to be expected in grading analyses.

## Moisture Determination

Moisture determinations were made on products only, as the feed was, in every case, so dry as to be dusty. In all cases, the weight of the solid plus water in the sample and the weight of the dry solid were determined by direct weighing and the percentage of moisture calculated from these figures.

## Power Measurements

Belt drive was used for the testing work. The power transmission is not so efficient, of course, as direct drive through herring-bone gears and the latter installations will give higher relative mechanical efficiencies than those recorded in this paper. The watt-hour meter and the voltmeter and ammeter were read at 5-min. intervals. The watt-hour meter readings are the basis for the figures of power consumption used, the readings of the indicating instruments being used for purposes of check only.

## Outline of Testing Work

### *Objects*

The specific object of the work described in the following pages was the determination of a set of constants and characteristic curves for the conical mill which could be applied to any installation. The 4½-ft. mill is large enough to do any class of work for which the conical mill is suited, its only limitation being a question of capacity. It was hoped to cover the question of variation in capacity due to variation in diameter by a few tests on mills of other sizes. It has, however, been impossible to do this, and the writer can offer but a tentative rule based on figures collected by correspondence.

### *Plan of Work*

The plan of the work was to start with some given set of conditions, for instance, a 16-in. cylindrical section 4006-lb. load of mixed balls, a trap rock feed of a given size, no moisture, mill level (Test 202); and, keeping these conditions constant, vary one other condition (Tests 203 and 204), in this case the feed rate, and determine the effect of this variation on the character of the product, the horsepower, and the relative mechanical efficiency. By varying in similar manner the size of the feed, the kind of rock fed, the percentage of moisture, length of cylindrical section, slope of the mill, and the character and weight of the crushing charge, the effects of such variations on the performance of the 4½-ft. mill were determined.

In the collection of the aforementioned data various attendant phenomena of considerable interest were observed. Thus the distribution in

the mill of the various sizes of balls composing a mixed charge was accurately determined, the effects of slope and moisture percentage on the possible maximum crushing charge were observed, and the lack of agreement between the feed and discharge rates over any short interval (5 min. for example), resulting in practice in a pulsation in the flow to subsequent machines in a mill flow sheet, was studied. These results are presented in their proper places later. A complete series of power tests, totaling more than 100, was made to afford a basis for a formula giving the horsepower required by a conical mill of any size.

## Power Tests

### *Description of Tests*

Six series of tests were made to determine variations in power consumption with varying conditions of loading, one series for each of the following conditions:

| Test Series No. | Length of Cylindrical Section, Inches | Condition of Pulp |
|---|---|---|
| 1 | 16 | Dry |
| 2 | 16 | Wet |
| 3 | 32 | Dry |
| 4 | 32 | Wet |
| 5 | 48 | Dry |
| 6 | 48 | Wet |

In each series the first set of power readings was made with the mill empty. Successive sets of readings were then taken with ball loads starting at 500 lb. (226.8 kg.) and increasing by 500-lb. steps. With each 500 lb. of balls, 170 lb. of trap rock was charged in order to prevent excessive wear and hammer in the mill. In the dry tests loading was continued until the surface of the load was considerably above the axis of the mill, discharge being prevented by plugging the discharge end. In the wet tests enough water was fed to produce a slight discharge throughout the series, and the tests were discontinued when discharge of balls commenced. Rock was fed in these latter tests from time to time to balance the rock carried off in the discharge, but no exact balance was attempted and the degree of balance attained is not known. The duration of the tests for each condition of loading varied from 30 to 90 min. Power readings were taken every 5 min. and the run was continued until the power-time curve became a horizontal line. The early readings in any given test were considerably higher than the last, due probably to cold bearings, slipping belts, etc., and were disregarded in making up the average power consumption for the run.

Fig. 2 is a graph covering all the power tests. The greatest variations from smooth curves occur near the end of the graphs for the 4½-ft. by 48-in. mill. These variations are due, in part at least, to an overloaded motor. Fig. 3 was plotted in an attempt to draw the curves for the different tests closely enough together to give a reasonable basis for an average curve upon which it would be possible to base an empirical form-

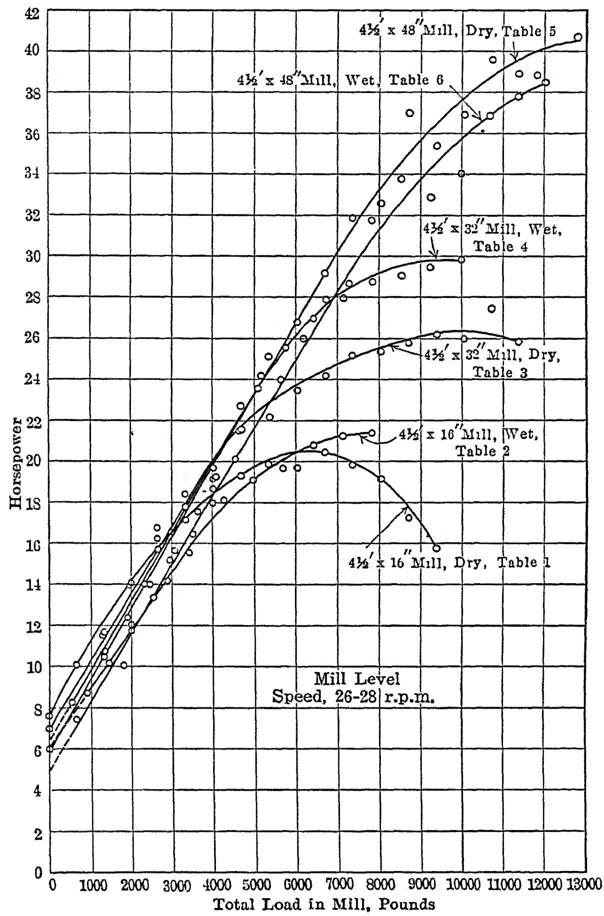



FIG. 2.—POWER CONSUMPTION OF 4½-FT. CONICAL BALL MILL AT HAMMOND LABORATORY.

ula for horsepower. As will be seen by reference to this figure, the curves are closely parallel throughout their respective lengths, the only graph departing seriously from parallelism with the others being that for the 4½-ft. by 16-in. mill, dry, series No. 1. It will be noted that the variation of this curve begins at the point where the charge in the mill rose above the horizontal axis. The mill was here working under unnatural

conditions, so that this variation will not affect a formula designed to cover working ranges only.

From this point two methods of procedure were followed, resulting in the following formulæ for the horsepower of a conical ball mill within working ranges.

$$(1)\ Hp. = 0.002(100D^2L^{0.132} + L) \qquad (1)$$

$$(2)\ Hp. = \frac{D^{2.42}L^{0.086}}{6.53} + \frac{L}{1000}(0.025m + 1.4) \qquad (2)$$

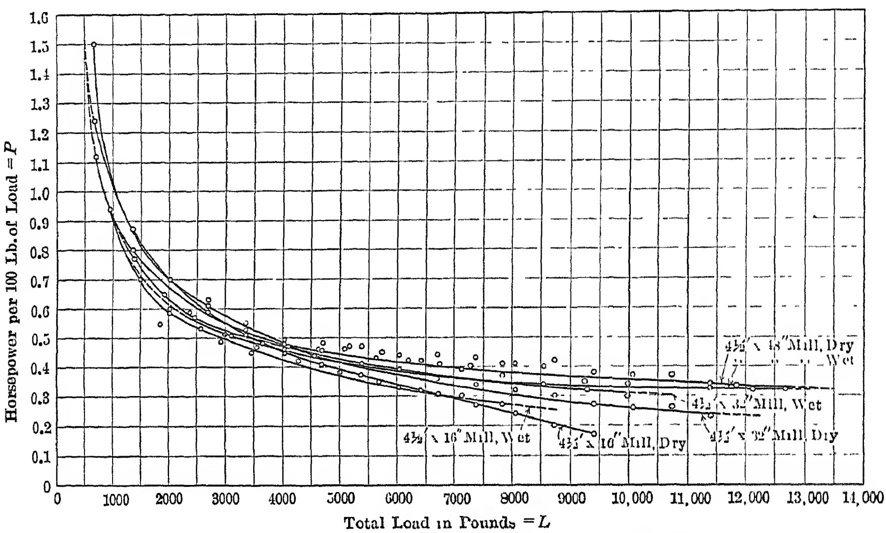



FIG 3.—POWER CONSUMPTION PER 100 LB. OF BALL LOAD BY 4½-FT. CONICAL BALL MILL WITH DIFFERENT CYLINDER LENGTHS.

where

$D$ = the internal diameter of the mill in feet,
$L$ = the total load in the mill in pounds,
$m$ = the length of the cylindrical section in inches.

The first of these formulæ is of the nature of a preliminary trial and was developed from a free-hand average curve drawn through the curves on Fig. 3. It does not, therefore, give results which check throughout the range of operating conditions. Formula (2) was developed as outlined in the succeeding paragraph.

The first step in the determination of Formula (2) was to plot the average curve shown in Fig. 4 from the curves on Fig. 3. The points determining this curve were obtained by averaging the ordinates of the curves on Fig. 3 at the abscissæ 1000, 2000, etc. Arbitrary ordinates $y$ and abscissæ $x$ were then assigned to this curve and various functions

such as $x^2$, $y^2$, $x^{1/2}$, $y^{1/2}$, $\log x$, $\log y$, $\frac{x}{y}$, $\frac{y}{x}$, $\frac{1}{x}$, $\frac{1}{y}$, etc., were plotted against each other in different combinations in an attempt to straighten out the curve. The best approximation to a straight line was obtained by plotting log $(y - 2)$ as ordinates and log $(x)$ as abscissæ. The points thus obtained are shown on Fig. 5. The straight line drawn through these points was obtained by averaging ordinates and abscissæ. The equation for this line is:

$$\log (y - 2) = 1.213 - 0.914 \log (x) \tag{4}$$

From (4)

$$\log (y - 2)(x^{0.914}) = 1.213 \tag{5}$$

Taking antilogarithms of both sides

$$(y - 2)(x^{0.914}) = 16.33 \tag{6}$$

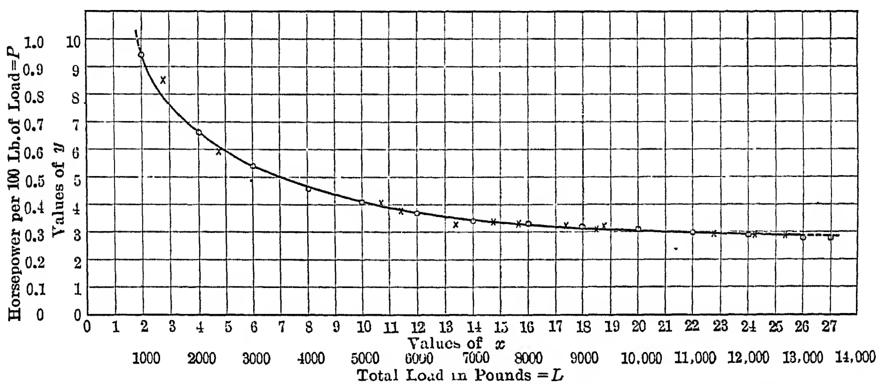



FIG. 4.—AVERAGE POWER CONSUMPTION OF $4\frac{1}{2}$-FT. CONICAL BALL MILL.

Clearing

$$y = \frac{16.33}{x^{0.914}} + 2 \tag{7}$$

But from Fig. 4

$$y = 10P \tag{8}$$

where $P$ = hp. per 100 lb. of load

$$x = \frac{L}{500} \tag{9}$$

Substituting these values for $x$ and $y$ in equation (7) and clearing

$$P = \frac{478}{L^{0.914}} + 0.2 \tag{10}$$

But

$$Hp. = \frac{LP}{100} \tag{11}$$

Then

$$Hp. = 4.78L^{0.086} + 0.002L \tag{12}$$

Horsepowers solved for by this formula for different loads are indicated by crosses ($x$) on Fig. 4. These check solutions show a close agreement with the average curve, but show in some cases as much as 30 per cent. departure from the horsepowers determined experimentally. The variations are, as might be expected, greatest for the 16-in. and 48-in. cylindrical sections, as the average curve of Fig. 4 departs most greatly from the curves for these cylinder lengths. In order to eliminate this variation, Formula (12) was written as follows:

$$Hp. = 4.78L^{0.086} + CL \tag{13}$$

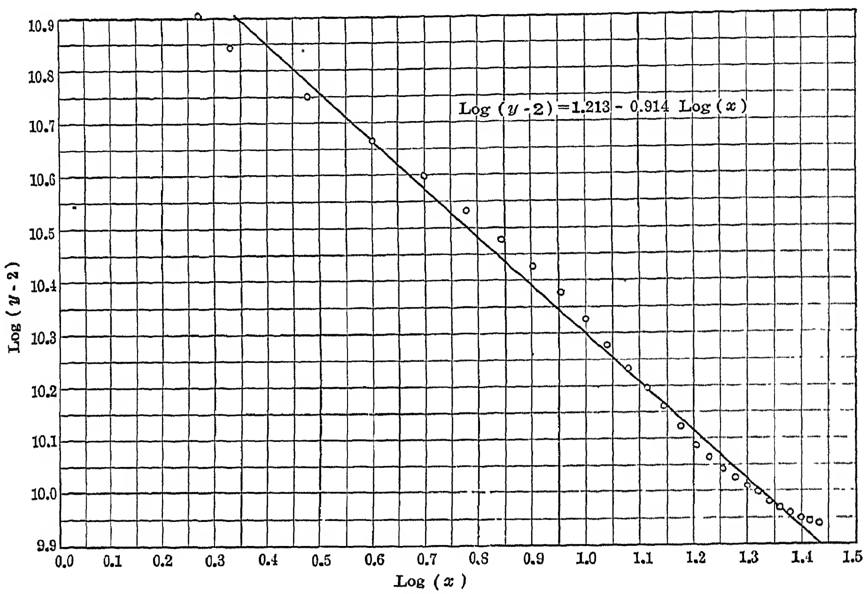



FIG. 5.

and average values of $C$ were determined for different cylinder lengths by substituting known values of $Hp.$ and $L$ corresponding to values of $m$ from results of tests, series 1 to 6 inclusive. By this method the following corresponding average values of $C$ and $m$ were determined:

| $m$ | $C$ |
|---|---|
| 16 | 0.0018 |
| 32 | 0.0022 |
| 48 | 0.0027 |

The relation between these quantities can be expressed in the linear form

$$C = 0.000025m + 0.0014 \tag{14}$$

from which equation (13) may be rewritten as

$$Hp. = 4.78L^{0.086} + \frac{L}{1000}(0.025m + 1.4) \tag{15}$$

which gives values for the horsepower of the 4½-ft. ball mill that are accurate within a few per cent. The average curve for horsepower for mills of other diameters plotted with horsepower per 100 lb. (45.36 kg.) of load as ordinates and total load as abscissæ will be similar to the curve

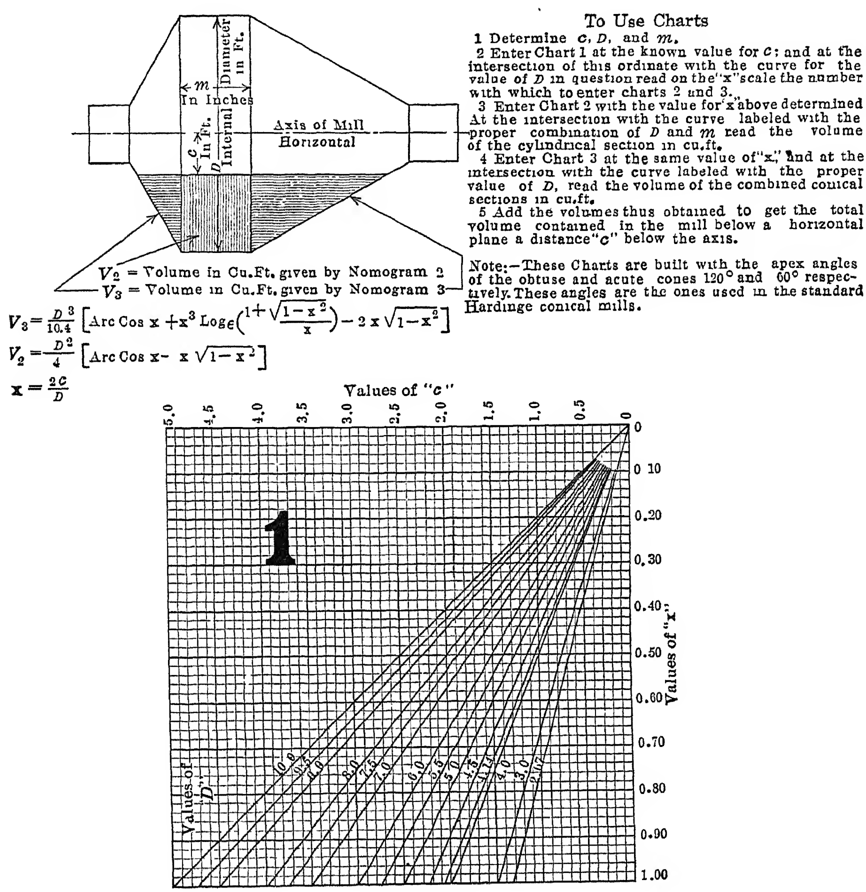
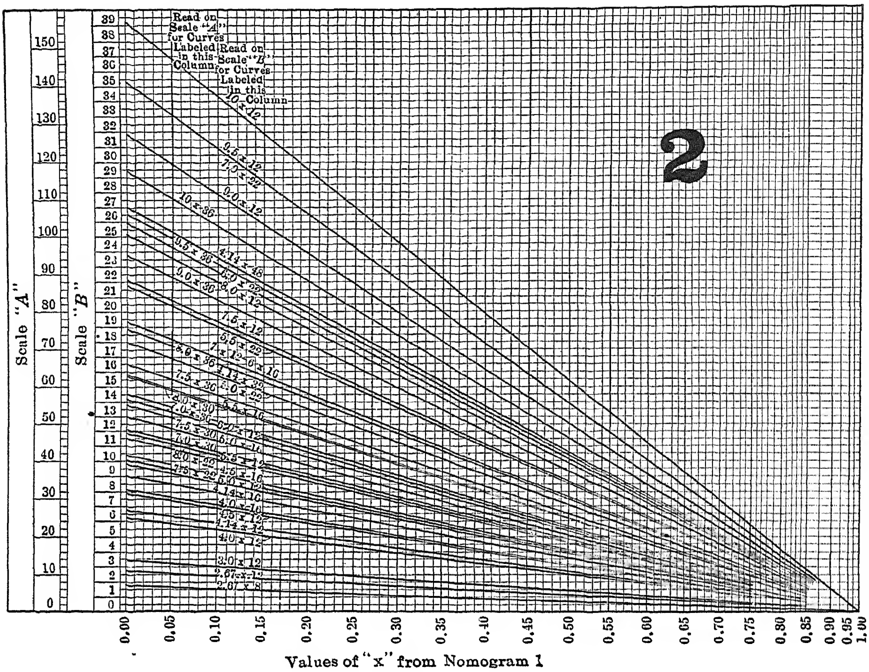



Fig. 6.

in Fig. 4 and, by changing the scale, can be made to coincide with this curve. If, then, corresponding values of $x$ and $L$ on this figure can be established for mills of several diameters and the proper substitutions made in equation (7) we will get a series of different numbers for the coefficient of the term $L^{0.086}$ in equation (15), corresponding to different internal diameters. The values of $L$ corresponding to a given value of



Scale "A"
Scale "B"
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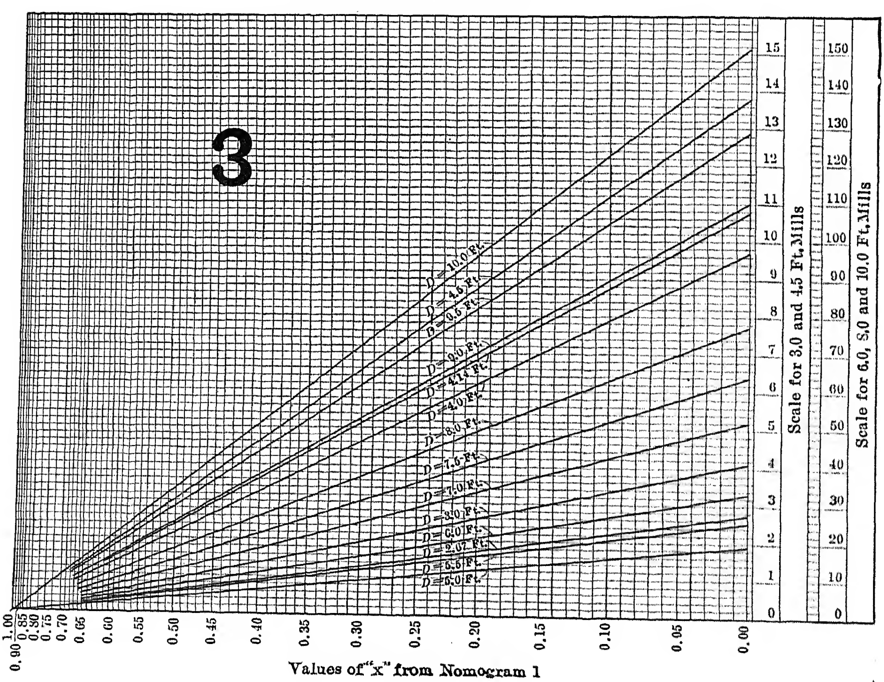
2
Values of "x" from Nomogram 1

3
Scale for 3.0 and 4.5 Ft. Mills
Scale for 6.0, 8.0 and 10.0 Ft. Mills
Values of "x" from Nomogram 1


FIG. 6.

$x$, Fig. 4, will vary according to the volumes of the different mills. These values and the corresponding values of the coefficients of $L^{0.086}$ are as follows:

| Internal Diameter of Mill in Feet | $x$ | $L$ | Coefficient of $L^{0.086}$ |
|---|---|---|---|
| 4.14 | 1 | 500 | 4.78 |
| 5.50 | 1 | 1,035 | 9.30 |
| 7.50 | 1 | 2,355 | 19.72 |
| 9.50 | 1 | 4,460 | 35.35 |

The logarithms of these coefficients of $L^{0.086}$ and the logarithms of the internal diameter of the mill, $D$, bear a linear relation to each other which is expressed in the equation

$$\log (D) = 0.413 \log (C) + 0.337 \tag{16}$$

From this equation

$$C = \frac{D^{2.42}}{6.53} = \text{coefficient of } L^{0.086} \tag{17}$$

Substituting this value for the coefficient of $L^{0.086}$ in equation (15) we have

$$Hp. = \frac{D^{2.42}L^{0.086}}{6.53} + \frac{L}{1000}(0.025m + 1.4) \tag{18}$$

This formula gives values accurate within a few per cent. for the horsepower of the conical ball mill throughout the range of operating conditions.

For pebble mills with smooth lining, results obtained by the above formula should be multiplied by the factor 0.65; with a semi-smooth lining, 0.8; with a rough lining, 0,95. It must be noted, however, in the use of the formula, that the load should be calculated on the assumption that the mill is horizontal, as the reduction in load due to tilting does not produce a corresponding decrease in power consumption.

The charts given in Fig. 6 and 7 will be found useful in determining the value of $L$ in the horsepower formula. The use of these charts may be best explained by following through a calculation for the horsepower consumed by an 8-ft. by 30-in. ball mill crushing rock of a specific gravity of 2.6 with a moisture content of 50 per cent., using a 30,000-lb. ball load, composed of 5-in., 4-in., and 3-in. balls. For this condition

$D = 7.5$.

$c$ (Fig. 6) $= 0$. (The volume contained in a mill in operation is more than that contained in the same mill at rest, and the assumption that $c = 0$ is legitimate.)

Then from nomogram 1,

$$x = 0.$$

Enter nomogram 3 with this value for $x$ and read on line $D = 7.5$, $V = 63.7$.

Enter nomogram 2 with $x = 0$ and read on line $7.5 \times 30$, $V = 55.3$.

The working volume of the mill is, then, $63.7 + 55.3 = 119.0$ cu. ft.

The volume occupied by the balls is $30{,}000 \div 495 = 60.6$ cu. ft.

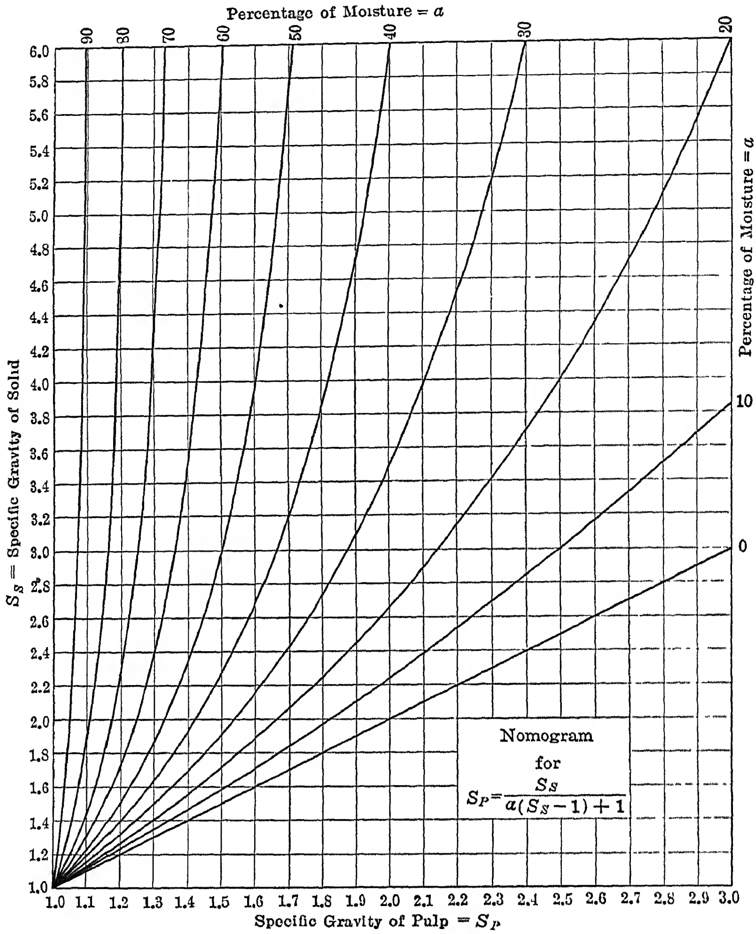



Fig. 7.

The volume occupied by the pulp is $119.0 - 60.6 = 58.4$ cu. ft.

The specific gravity of the pulp is determined from Fig. 7.

Enter at $S_S = 2.6$. At the intersection with the curve $a = 50$ per cent., read $S_P = 1.44$. The weight of the pulp in the mill is, then, $58.4(1.44)62.5 = 5250$ lb.

$$L = 30{,}000 + 5{,}250 = 35{,}250.$$

$$Hp. = \frac{(7.5^{2.42})(35{,}250)^{0.086}}{6.53} + \frac{35{,}250}{1{,}000}(0.025(30) + 1.4)$$
$$= 125.2.$$

The economy in calculation to be gained from the use of Fig. 6 is not so apparent in the foregoing instance, where $c$ was taken equal to zero, as it will be if the information sought is the crushing load which fills a given mill to within a given distance of the center, or the depth to which a given load will fill a mill of a given size. In the course of 2 or 3 years' work with the mill, the writer has been confronted with a consider-

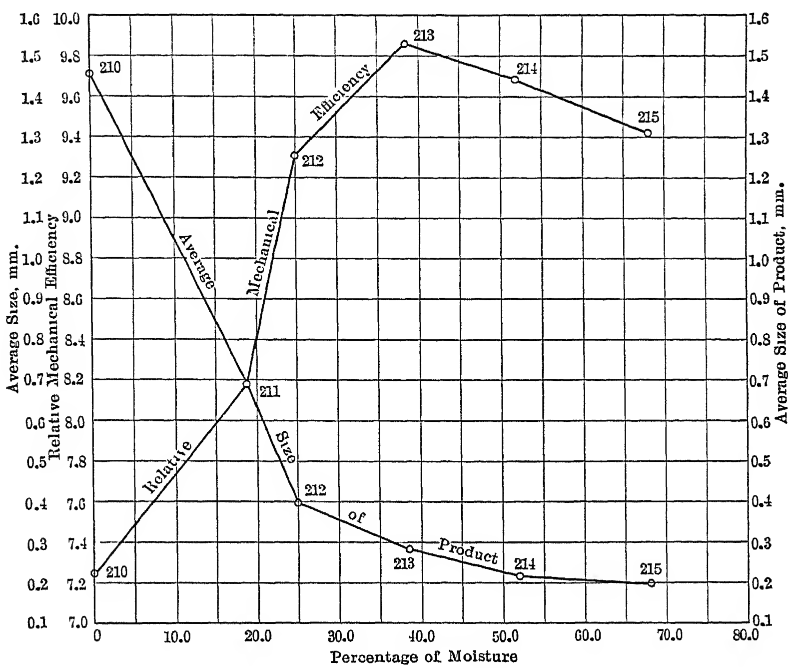



FIG. 8.—EFFECT OF MOISTURE ON CRUSHING EFFICIENCY AND AVERAGE SIZE OF PRODUCT.

able number of such problems and it is because of the saving in time effected in their solution by the use of the chart, that it is inserted here. In such calculations the weight of a cubic foot of steel balls may be taken as 250 lb. and the weight of a cubic foot of pebbles as 100 lb.

## ANALYSIS OF OPERATING DATA

Thirty-five tests were run on the 4½-ft. mill to determine the effect of variations in operating conditions on the performance of the mill. As each test furnished some information that may be classified under several heads, it is not possible, without considerable repetition, to segre-

gate them. They are, therefore, presented in Tables 1 and 2 in the order in which they were performed and will be referred to by number in the subsequent discussion.

The indicator commonly used in this paper for comparing the character of the work done by the mill under any given condition with that done under some other condition is the figure in the last column of Table 1 headed R.M.E. (Relative Mechanical Efficiency). A detailed explanation of the development of this conception is given in the article,

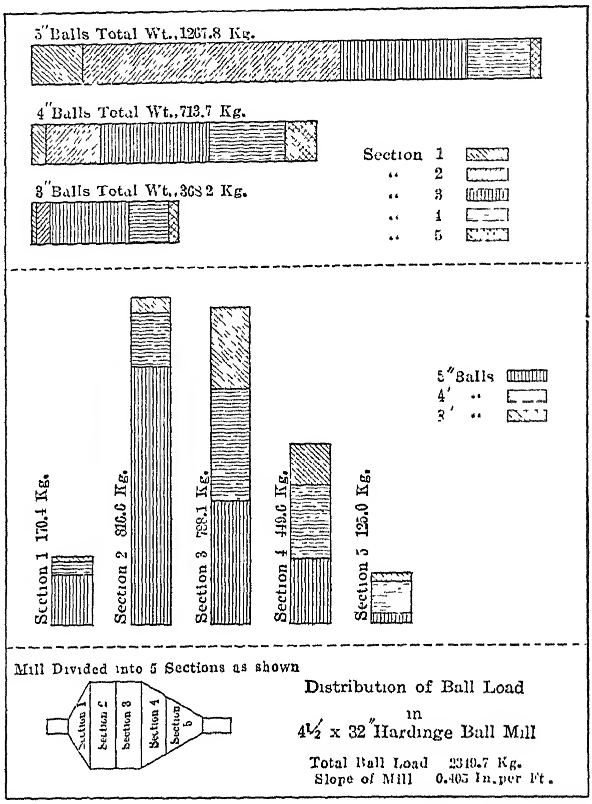



Fig. 9.

The Work of Crushing, *Trans.* (1914), **48,** 153. Briefly, it is expressed in the formula

$$\text{R.M.E} = \frac{\text{(Difference E.U. Feed and Product) (Tons per 24 hr.)}}{\text{Hp.}}$$

in which the term "Difference E.U. Feed and Product" is a measure of the useful work done per unit of weight in reducing the material in question from feed size to discharge size, and is determined by screen analysis.

Thus the total energy units, E.U., in the feed sample, screen test 22-A, Table 2, is 34.32 and is obtained by summing the products of the percentages on the different screens by their corresponding ordinal numbers. The same method applied to screen test 22–41 gives 1664.22 E.U. in the product of run No. 202. Then

$$\text{R.M.E.} = \frac{(1664.22 - 34.32)12}{21} = 930.$$

In order to make this figure accord with commonly accepted figures of efficiency, the R.M.E. thus obtained is divided by 100, giving for test 202 a value of 9.3.

*Rate of Feed*

The effect of rate of feed on the relative mechanical efficiency of the conical ball mill is given in the two groups of tests 202 to 204 and 213, 216 to 219. In the first group, trap rock of an average size of 24.58 mm. (0.96 in.) was fed dry at the rates of 1000 lb. (453.59 kg.), 1500 lb., and 2000 lb. per hour. The axis of the mill was horizontal and the ball load was a mixture of 5-in., 4-in., 3-in., and 1¾-in. balls in approximately the same proportions that would be found in commercial operation after the mill had settled down. The relative mechanical efficiencies, 9.3, 9.13 and 10.3 respectively indicate the result, confirmed in later tests, that the ratio of useful work done by the mill to power input increases with the feed rate. That there is, of course, a limit to this proportionate increase at the point of overload is shown in the second series of tests above mentioned. In this series the mill was tilted 2⅝ in., or 0.405 in. per foot, toward the discharge end. One result of this tilting was to decrease the ball capacity of the mill by about 1,200 lb. Quartzite of an average size of 9.90 mm. was fed with an average of about 38 per cent. moisture at rates of 1500 lb., 3000 lb., 6000 lb., 9000 lb. and 12,000 lb. per hour. The relative mechanical efficiencies corresponding to the above rates were 9.86, 17.30, 29.21, 43.50, and 41.10. In this series of tests the relative mechanical efficiency of the machine increases with the feed rate up to 4.5 tons per hour, beyond which we have an apparent condition of overloading. Table 3 gives the reduction in average size of particle in the different tests above discussed.

These figures present three different cases for consideration. Test 202 is in a class by itself, the machine is patently underfed for all purposes except that of producing a practically finished, fine, dry product at one passage through the machine. It is a surprising fact that in doing this kind of work the machine uses power so efficiently. Tests 203, 204, 218 and 219 compared with tests 213, 216 and 217 point the moral that for most efficient work it is not wise to attempt too great reduction at one passage through the mill. When the large amount of power consumed

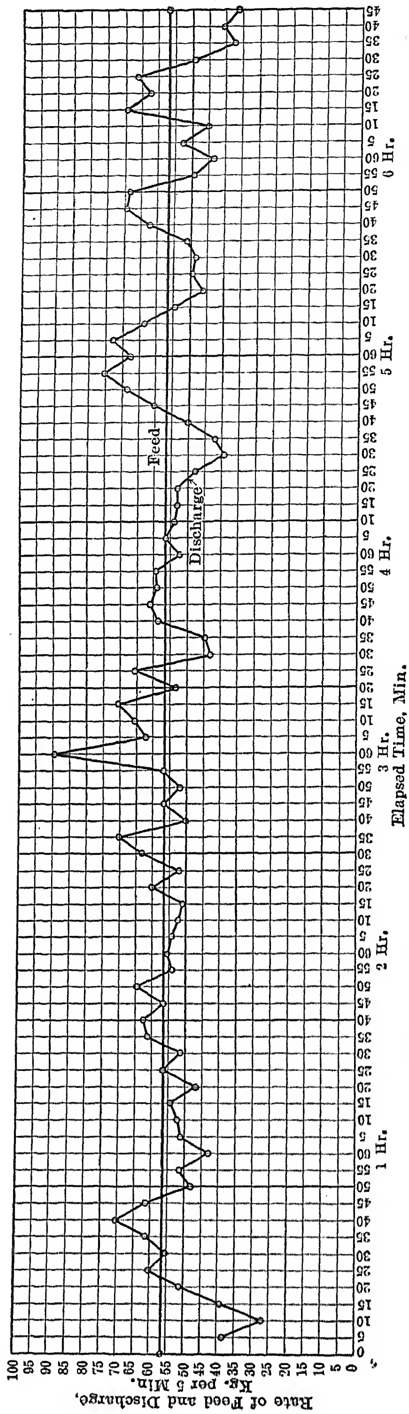



FIG. 10.—COMPARISON OF FEED AND DISCHARGE RATES OF CONICAL BALL MILL DRY CRUSHING.

TABLE 1

| Test No. | Size of Mill | Crushing Charge | | | | | | Kind of Rock Crushed | Screen Test No | | Feed Rate, Pounds per Hour | Percentage of Moisture | Elevation of Feed End, Inches | R p m | Hp. | R m e. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Character | Weight, Pounds | | | | | | Feed | Product | | | | | | |
| | | | 5-in | 4-in | 3-in | 1¾-in | Total | | | | | | | | | |
| 202 | 4½ by 16 | Balls | 1,960 | 953 | 712 | 381 | 4,006 | Trap | 22–A | 22–41 | 1,000 | 0.0 | 0 | 28.0 | 21.0 | 9.30 |
| 203 | 4½ by 16 | Balls | 1,960 | 953 | 712 | 381 | 4,006 | Trap | 22–A | 22–69 | 2,000 | 0.0 | 0 | 28.0 | 21.0 | 10.30 |
| 204 | 4½ by 16 | Balls | 1,960 | 953 | 712 | 381 | 4,006 | Trap | 22–A | 22–77 | 1,500 | 0.0 | 0 | 28.0 | 21.3 | 9.13 |
| 205 | 4½ by 16 | Balls | 2,455 | 1,110 | 703 | 235 | 4,503 | Trap | 22–A | 22–79 | 1,500 | 0.0 | 0 | 28.0 | 22.1 | 11.03 |
| 206 | 4½ by 16 | Balls | 2,400 | 522 | ..... | .... | 2,922 | Trap | 22–A | 22–88 | 1,500 | 0.0 | 4¼ | 26.5 | 23.0 | 8.66 |
| 207 | 4½ by 16 | Balls | 2,400 | 522 | ..... | .... | 2,922 | Trap | 22–A | 22–91 | 1,500 | 0.0 | 2⅝ | 28.0 | 22.2 | 9.33 |
| 208 | 4½ by 16 | Balls | 2,400 | 522 | ..... | . | 2,922 | Trap | 22–A | 38–4 | 1,500 | 19.5 | 2⅝ | 26.5 | 19.9 | 11.46 |
| 209 | 4½ by 16 | Balls | 2,400 | 522 | .... | ..... | 2,922 | Quartzite | 39–A | 39–3 | 1,500 | 0.0 | 2⅝ | 25.5 | 18.4 | 8.09 |
| 210 | 4½ by 16 | Balls | 1,460 | 885 | 385 | ..... | 2,930 | Quartzite | 39–A | 40–1 | 1,500 | 0.0 | 2⅝ | 26.0 | 19.2 | 7.25 |
| 211 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 41A–1 | 1,500 | 18.8 | 2⅝ | 27.0 | 17.7 | 8.18 |
| 212 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 41B–1 | 1,500 | 25.0 | 2⅝ | 27.0 | 17.7 | 9.31 |
| 213 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 41C–1 | 1,500 | 38.5 | 2⅝ | 27.0 | 17.7 | 9.86 |
| 214 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 41D–1 | 1,500 | 51.8 | 2⅝ | 26.0 | 19.3 | 9.67 |
| 215 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 41E–1 | 1,500 | 68.2 | 2⅝ | 26.0 | 20.1 | 9.42 |
| 216 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 42–1 | 3,000 | 37.4 | 2⅝ | 26.0 | 18.5 | 17.30 |
| 217 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 42–2 | 6,000 | 37.3 | 2⅝ | 26.0 | 17.7 | 29.21 |
| 218 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 42–3 | 12,000 | 38.3 | 2⅝ | 26.0 | 18.5 | 41.10 |
| 219 | 4½ by 16 | Balls | 1,460 | 878 | 481 | ..... | 2,819 | Quartzite | 39–A | 42–4 | 9,000 | 38.9 | 2⅝ | 26.0 | 16.9 | 43.50 |
| 220 | 4½ by 16 | Balls | 1,772 | 1,067 | 711 | ..... | 3,550 | Quartzite | 39–A | 44–1 | 1,500 | 31.7 | 1¼ | 26.5 | 20.1 | 8.45 |
| 221 | 4½ by 16 | Balls | 2,125 | 1,284 | 855 | ..... | 4,264 | Quartzite | 39–A | 44–2 | 1,500 | 36.5 | 0 | 26.0 | 20.1 | 9.06 |
| 222 | 4½ by 16 | Balls | 2,125 | 1,284 | 855 | ..... | 4,264 | Trap | 45–A | 45–1 | 1,500 | 35.4 | 0 | 25.0 | 20.9 | 10.20 |
| 223 | 4½ by 16 | Balls | 1,457 | 874 | 480 | ..... | 2,811 | Trap | 45–A | 46–1 | 1,500 | 41.7 | 2⅝ | 26.5 | 20.1 | 10.71 |
| 224 | 4½ by 16 | Balls | 696 | 417 | 293 | ..... | 1,406 | Quartzite | 39–A | 47–1 | 1,500 | 47.0 | 2⅝ | 26.5 | 15.3 | 9.83 |
| 225 | 4½ by 16 | Balls | 1,412 | .... | ..... | ..... | 1,412 | Quartzite | 39–A | 47–2 | 1,500 | 40.0 | 2⅝ | 27.0 | 13.7 | 10.72 |
| 226 | 4½ by 32 | Balls | 2,785 | 1,572 | 810 | ..... | 5,167 | Quartzite | 39–A | 49–1 | 1,500 | 39.4 | 3 3/16 | 25.0 | 26.6 | 7.15 |
| 227 | 4½ by 48 | Balls | 3,765 | 1,822 | 1,067 | ..... | 6,654 | Quartzite | 39–A | 50–1 | 1,500 | 40.4 | 0.405 in./ft. | 24.5 | 33.5 | 5.88 |
| 228 | 4½ by 48 | Pebbles | ..... | .... | 3,125 | ..... | 3,125 | Quartzite | 51–A | 51–1 | 1,500 | 42.2 | 0 | 27.5 | 16.9 | 5.37 |
| 229 | 4½ by 48 | Pebbles | ..... | .... | 3,125 | ..... | 3,125 | Trap | 52–A | 52–1 | 1,500 | 37.6 | 0 | 26.5 | 16.4 | 1.89 |
| 230 | 4½ by 48 | Balls | 1,604 | 1,479 | 2,160 | 70 | 5,313 | Cocoanut shell | 32–1 | 32–2 | 1,000 | 0.0 | 2⅝ | 25.0 | 22.1 | 1.67 |
| 231 | 4½ by 48 | Balls | 343 | 1,480 | 2,720 | 758 | 5,301 | Cocoanut shell | 32–2a | 32–3 | 550 | 0.0 | 2⅝ | 28.0 | 23.6 | 0.50 |
| 232 | 4½ by 48 | Balls | 343 | 1,480 | 2,720 | 500 | 5,043 | Cocoanut shell | 32–4 | 32–7 | 550 | 84.5 | 0 | 28.0 | 31.4 | 1.56 |
| 233 | 4½ by 48 | Balls | 343 | 1,480 | 2,720 | 500 | 5,043 | Cocoanut shell | 32–6 | 32–8 | 2,200 | 54.8 | 0 | 27.0 | 32.8 | 0.94 |
| 234 | 4½ by 16 | Balls | ..... | 1,465 | 2,535 | ..... | 4,000 | Sawdust | 5,321 | 5,331 | 300 | 0.0 | 0 | 27.0 | 15.5 | 0.48 |
| 235 | 4½ by 16 | Balls | 1,927 | 951 | 815 | 720 | 4,413 | Brass ashes | 1 | 2 | 4,480 | 42.9 | 2½ | 28.0 | 20.0 | 15.60 |
| 236 | 4½ by 16 | Balls | 1,927 | 951 | 815 | 720 | 4,413 | Brass ashes | 3 | 4 | 1,820 | 39.7 | 2½ | 28.0 | 21.0 | 10.30 |

### TABLE 2.—*Screen Tests*

| Ordinal Number | Screen Aperture, Millimeters | 22–A | 22–41 | 22–69 | 22–77 | 22–79 | 22–88 | 22–91 |
|---|---|---|---|---|---|---|---|---|
| −1.0 | 38.100 | 2 15 | | | | | | |
| 0 | 26.670 | 70.90 | | 11 72 | 7 41 | 6 67 | 5 05 | 1 13 |
| 1 | 18.850 | 24.42 | | 16.84 | 14 78 | 5 78 | 6 10 | 3 87 |
| 2 | 13.330 | 1 63 | | 11 62 | 9 27 | 2 28 | 5 95 | 5 16 |
| 3 | 9.423 | 0 23 | | 4 60 | 3 84 | 0 83 | 4 98 | 5 44 |
| 4 | 6 680 | 0 10 | . . | 2 46 | 1 42 | 0 46 | 3 60 | 4 99 |
| 5 | 4 699 | 0 05 | . . | 1.21 | 0 49 | 0 32 | 3 28 | 5 44 |
| 6 | 3.327 | 0.04 | . . | 0 50 | 0.38 | 0 50 | 2 60 | 4 48 |
| 7 | 2.362 | 0 02 | 0 17 | 0 36 | 0 29 | 0 32 | 2 62 | 3 00 |
| 8 | 1.651 | 0 02 | 0 85 | 0.28 | 0 37 | 0 50 | 2 54 | 3.09 |
| 9 | 1.168 | 0.02 | 0 48 | 0 37 | 0 40 | 0 86 | 2.72 | 2.71 |
| 10 | 0 833 | 0 02 | 0 92 | 0.46 | 0 52 | 1 47 | 2 58 | 2 35 |
| 11 | 0 589 | 0 02 | 2 49 | 0 79 | 1 10 | 2 92 | 3 02 | 2.50 |
| 12 | 0.417 | 0 02 | 3.57 | 1.21 | 1.39 | 3 71 | 2 82 | 2 34 |
| 13 | 0 295 | 0 03 | 6.10 | 2.20 | 2 74 | 5 15 | 3 24 | 2 95 |
| 14 | 0 208 | 0 04 | 5 98 | 2 97 | 3 63 | 5 88 | 3 50 | 3 17 |
| 15 | 0.147 | 0.04 | 9.05 | 4.78 | 5 50 | 6 94 | 5 04 | 5 52 |
| 16 | 0.104 | 0.05 | 11 75 | 6 56 | 7 55 | 9 39 | 6 41 | 6 49 |
| 17 | 0 074 | 0.04 | 8 34 | 4.92 | 6 12 | 7.47 | 5.60 | 5 75 |
| 19 | Through 0 074 | 0.16 | 50 30 | 26.15 | 32 80 | 38.55 | 28.35 | 29 62 |
| | Aver size of particle, mm .... | 24.58 | 0 142 | 8 590 | 6 573 | 3.449 | 4.472 | 3.219 |
| | Total energy units | 34.32 | 1,664 22 | 936.00 | 1,114.80 | 1,387.64 | 1,138 47 | 1,183 64 |

### TABLE 2.—*Screen Tests.—(Continued)*

| Ordinal Number | Screen Aperture, Millimeters | 38–4 | 39–A | 39–3 | 40–1 | 41A–1 | 41B–1 | 41C–1 |
|---|---|---|---|---|---|---|---|---|
| −1.0 | 38.100 | ....... | 0.66 | | | | | |
| 0 | 26 670 | 4 34 | 5.51 | 0.22 | 0.45 | | | |
| 1 | 18.850 | 3.22 | 16.36 | 0.76 | 1.44 | 0.03 | | |
| 2 | 13.330 | 2.98 | 22.00 | 1.18 | 2.02 | 0.25 | | |
| 3 | 9.423 | 2.32 | 11.18 | 1.00 | 1.63 | 0.45 | ......... | 0.01 |
| 4 | 6.680 | 2.30 | 7.46 | 1.16 | 1.37 | 0.72 | 0.11 | 0.02 |
| 5 | 4.699 | 1.82 | 5.49 | 1.38 | 1.83 | 0.70 | 0.28 | |
| 6 | 3.327 | 1 72 | 3.62 | 1.54 | 2.44 | 1.42 | 0.58 | 0.14 |
| 7 | 2 362 | 1.41 | 2.97 | 2.63 | 2.20 | 2.56 | 0.88 | 0.26 |
| 8 | 1.651 | 1.90 | 3.30 | 4.45 | 3.78 | 5.01 | 2.38 | 1.07 |
| 9 | 1.168 | 1.94 | 2.94 | 5 39 | 4.96 | 6.94 | 3.99 | 2.14 |
| 10 | 0.833 | 2.89 | 3.18 | 6.04 | 5.92 | 8.64 | 6.73 | 4.33 |
| 11 | 0.589 | 3.93 | 3.55 | 8 47 | 9.38 | 11.94 | 11.88 | 9.13 |
| 12 | 0.417 | 4.66 | 2.37 | 7.46 | 7.01 | 9.14 | 9.14 | 10.05 |
| 13 | 0.295 | 5.49 | 2.48 | 8.31 | 9.58 | 10.22 | 11.27 | 13.10 |
| 14 | 0.208 | 6.29 | 1.85 | 7.46 | 8.36 | 7.61 | 8.91 | 10.87 |
| 15 | 0.147 | 8.91 | 1.72 | 10.13 | 9.62 | 8.99 | 9.93 | 11.59 |
| 16 | 0.104 | 9.15 | 1.25 | 8.35 | 7.67 | 6 62 | 8.23 | 9.76 |
| 17 | 0.074 | 6.43 | 0.70 | 5.74 | 4.89 | 4.21 | 5.30 | 5.89 |
| 19 | Through 0 074 | 28.30 | 1.41 | 18.33 | 15.45 | 14.55 | 20.39 | 21.64 |
| | Aver. size of particle, mm. | 2.896 | 9.900 | 1.052 | 1.456 | 0.688 | 0.397 | 0.288 |
| | Total energy units | 1,301.83 | 487.82 | 1,316.38 | 1,261.70 | 1,292.85 | 1,403.48 | 1,458.70 |

TABLE 2.—*Screen Tests.*—(*Continued*)

| Ordinal Number | Screen Aperture, Millimeters | 41D–1 | 41E–1 | 42–1 | 42–2 | 42–3 | 42–4 | 44–1 |
|---|---|---|---|---|---|---|---|---|
| −1.0 | 38.100 | | | | | | | |
| 0 | 26.670 | ...... | ...... | . .. | ... . | 0.08 | | |
| 1 | 18.850 | . .. . | . .... | ... .. | ... | 0.76 | 0.58 | |
| 2 | 13.330 | ...... | .. ... | ...... | ..... | 2.99 | 1.44 | |
| 3 | 9.423 | . . | .. ... | 0.02 | 0.04 | 3.38 | 2.55 | |
| 4 | 6 680 | . | ... | 0 02 | 0 29 | 4.08 | 3.59 | 0.02 |
| 5 | 4 699 | | | ... . | 1.08 | 4.03 | 2.64 | |
| 6 | 3 327 | 0.11 | | 0 29 | 2 47 | 5.40 | 3.67 | 0.16 |
| 7 | 2 362 | 0.05 | 0.06 | 0.83 | 4 56 | 6.96 | 5.96 | 0.41 |
| 8 | 1 651 | 0.33 | 0.15 | 2 45 | 8.22 | 8.70 | 8.02 | 1.27 |
| 9 | 1.168 | 0.75 | 0.48 | 4 34 | 9.30 | 8 06 | 8.10 | 2.76 |
| 10 | 0 833 | 1 93 | 1 58 | 6 92 | 9.86 | 7.79 | 8.80 | 5 41 |
| 11 | 0 589 | 6.22 | 4.68 | 12.37 | 12 35 | 9 88 | 11.37 | 10.50 |
| 12 | 0.417 | 8.91 | 7.98 | 10 15 | 9.18 | 6.99 | 8.40 | 10.96 |
| 13 | 0 295 | 12.20 | 12.57 | 12.40 | 9 72 | 7.73 | 8.42 | 12.45 |
| 14 | 0 208 | 11.27 | 12.48 | 10 55 | 6 91 | 5 91 | 6.57 | 9.35 |
| 15 | 0.147 | 13 21 | 14.73 | 10 88 | 8.10 | 5.93 | 6.74 | 11.73 |
| 16 | 0 104 | 11.50 | 11.87 | 8 48 | 5 72 | 4 24 | 4.84 | 9.37 |
| 17 | 0 074 | 7.29 | 7.35 | 5.12 | 3 78 | 2.44 | 2.84 | 6.14 |
| 19 | Through 0 074 | 26.23 | 26 07 | 15.18 | 8 42 | 4.65 | 5.47 | 19.47 |
| | Aver size of particle, mm .. | 0 215 | 0.196 | 0 388 | 0 768 | 2.129 | 1.628 | 0.316 |
| | Total energy units | 1,525 87 | 1,538 26 | 1,376.66 | 1,207.65 | 1,016 92 | 1,169.53 | 1,432.96 |

TABLE 2.—*Screen Tests.*—(*Continued*)

| Ordinal Number | Screen Aperture Millimeters, | 44–2 | 45–A | 45–1 | 46–1 | 47–1 | 47–2 | 49–1 |
|---|---|---|---|---|---|---|---|---|
| −1.0 | 38.100 | | | | | | | |
| 0 | 26 670 | ....... | 5 62 | | | | | |
| 1 | 18 850 | ....... | 14 72 | | | | | |
| 2 | 13 330 | ...... | 24 42 | | | | | |
| 3 | 9.423 | ...... | 15 92 | | | | | |
| 4 | 6.680 | 0.02 | 10 18 | 0 04 | 0.01 | 0.02 | 0 04 | 0.01 |
| 5 | 4.699 | 0 07 | 7 65 | 0 38 | ....... | ........ | 0.53 | |
| 6 | 3.327 | 0.08 | 4 92 | 0.08 | ... ... | 0 78 | 1.21 | |
| 7 | 2.362 | 0.27 | 3.03 | 0.36 | 0.52 | 1.79 | 2.24 | 0.04 |
| 8 | 1 651 | 0 56 | 2.50 | 1.27 | 1.12 | 5 44 | 5.30 | 0.41 |
| 9 | 1.168 | 1 29 | 1.71 | 2.05 | 2 24 | 7.98 | 8.20 | 0.89 |
| 10 | 0.833 | 3.12 | 1.25 | 4 49 | 3.15 | 9.07 | 9.04 | 1.93 |
| 11 | 0.589 | 7.10 | 1 19 | 7.58 | 6.26 | 11.77 | 11.53 | 5.07 |
| 12 | 0.417 | 8.71 | 0.79 | 7.82 | 6 84 | 8.30 | 8.65 | 7.20 |
| 13 | 0.295 | 12.29 | 0.83 | 8.88 | 7.66 | 9.87 | 9.79 | 11.80 |
| 14 | 0.208 | 11.33 | 0.57 | 7.29 | 6 62 | 7 91 | 8.10 | 11.41 |
| 15 | 0.147 | 13.17 | 0 84 | 9.97 | 8.58 | 9.16 | 9 04 | 13.75 |
| 16 | 0.104 | 10.59 | 0.83 | 9.84 | 8.81 | 7 99 | 7.98 | 11.29 |
| 17 | 0.074 | 6.98 | 0.68 | 7.03 | 6 32 | 5.44 | 5.23 | 8.72 |
| 19 | Through 0.074 | 24.42 | 2 35 | 32.92 | 41.87 | 14.54 | 13.12 | 27.48 |
| | Aver. size of particle, mm ...... | 0.246 | 10.394 | 0.275 | 0.228 | 0.509 | 0.537 | 0.202 |
| | Total energy units | 1,500.73 | 384.29 | 1,525.27 | 1,580.21 | 1,324.53 | 1,305.35 | 1,543.11 |

TABLE 2.—*Screen Tests.*—(*Continued*)

| Ordinal Number | Screen Aperture, Millimeters | 50–1 | 51–A | 51–1 | 52–A | 52–1 | 32–1 | 32–2 |
|---|---|---|---|---|---|---|---|---|
| −1.0 | 38.100 | . . | . . . | ... .. | ...... | . . | 11.82 | |
| 0 | 26.670 | . | | ... . | . .. | . | 45 47 | 2 88 |
| 1 | 18.850 | . | 0 14 | ...... | ...... | ... | 20.97 | 12 58 |
| 2 | 13 330 | . . | 0 39 | ... .. | 0 02 | | 12 67 | 25 14 |
| 3 | 9.423 | . . | 0 76 | ...... | 0.06 | . . | 3 96 | 21 01 |
| 4 | 6 680 | 0 02 | 1.19 | 0 02 | 0 18 | | 1 88 | 15 26 |
| 5 | 4 699 | . . | 3.60 | ...... | 1 78 | . | 0 65 | 6 46 |
| 6 | 3 327 | . ... | 5.15 | 0 36 | 1.33 | . . | 0 41 | 3 34 |
| 7 | 2 362 | 0 04 | 5 44 | .. ... | 1.50 | 0.06 | 0 48 | 2 02 |
| 8 | 1 651 | 0 43 | 8 47 | 0 25 | 2.31 | 0.12 | 0 22 | 1 33 |
| 9 | 1.168 | 0.60 | 8 47 | 0 32 | 2.20 | 0.23 | 0 22 | 0 94 |
| 10 | 0.833 | 1 34 | 8 79 | 0 38 | 2 43 | 0.52 | 0 22 | 0 75 |
| 11 | 0 589 | 3 89 | 11 01 | 1 08 | 3 60 | 0.67 | 0 25 | 1.37 |
| 12 | 0 417 | 5 89 | 8.66 | 2.12 | 3 79 | 0.58 | 0 19 | 1 15 |
| 13 | 0.295 | 10.76 | 9 06 | 5 29 | 5 79 | 1.42 | 0.15 | 1.14 |
| 14 | 0.208 | 11.65 | 6 82 | 9.78 | 5 71 | 3 48 | 0 09 | 0 88 |
| 15 | 0.147 | 14 90 | 7.49 | 17 18 | 9 69 | 11 44 | 0 09 | 0 76 |
| 16 | 0 104 | 11.58 | 5.10 | 17.65 | 9 63 | 15.41 | 0.11 | 0 92 |
| 17 | 0.074 | 9 69 | 2.93 | 12 26 | 8.37 | 13 60 | 0.05 | 0.62 |
| 19 | Through 0 074 | 29 21 | 6.53 | 33.31 | 41.61 | 52.47 | 0.10 | 1 45 |
| | Aver size of particle, mm. .. . | 0.184 | 1.173 | 0 140 | 0.380 | 0.091 | 22.820 | 10.010 |
| | Total energy units | 1,583 89 | 1,130.23 | 1,635 04 | 1,563.10 | 1,735 92 | 83.00 | 390 15 |

TABLE 2.—*Screen Tests.*—(*Continued*)

| Ordinal Number | Screen Aperture, Millimeters | 32–2a | 32–3 | 32–4 | 32–7 | 32–6 | 32–8 | 5321 |
|---|---|---|---|---|---|---|---|---|
| −1 0 | 38 100 | | | | | | | |
| 0 | 26 670 | 4.67 | 0.09 | ....... | | | | |
| 1 | 18.850 | 20 41 | 6 22 | | | | | |
| 2 | 13.330 | 40.80 | 27.25 | ....... | ...... | 6 00 | 0.80 | |
| 3 | 9.423 | 34 12 | 34.03 | 15.41 | ...... | 15.89 | 3.85 | |
| 4 | 6 680 | ..... .. | 18.23 | 27.77 | 0.20 | 23.27 | 13.32 | 0.13 |
| 5 | 4.699 | ....... | 5.35 | 20.86 | 0.81 | 12.88 | 9.79 | 0.60 |
| 6 | 3.327 | ....... | 2.15 | 16 75 | 1.18 | 14.72 | 13.83 | 2.10 |
| 7 | 2.362 | ... ... | 0 69 | 8.23 | 2.31 | 8.91 | 7.99 | 5.81 |
| 8 | 1.651 | ....... | 0.17 | 4.40 | 3.85 | 4.19 | 8.01 | 19.42 |
| 9 | 1.168 | ...... | 0 07 | 2.39 | 6.11 | 2.13 | 6.03 | 23.27 |
| 10 | 0 833 | . . .. | 0.07 | 1.34 | 9.38 | 1.53 | 5.70 | 25.11 |
| 11 | 0 589 | ... ... | 0.09 | 1 15 | 14 94 | 1.39 | 7.46 | 17.89 |
| 12 | 0.417 | ....... | 0 11 | 0.33 | 14.78 | 0.86 | 4.71 | 4.33 |
| 13 | 0 295 | ....... | 0.18 | 0 43 | 12.63 | 0.84 | 4.63 | 0.80 |
| 14 | 0 208 | .. .... | 0.18 | 0.24 | 8 01 | 0.57 | 2.79 | 0.18 |
| 15 | 0 147 | ..... | 0.36 | 0.14 | 6.36 | 0.73 | 2.49 | 0.12 |
| 16 | 0.104 | .. . . | 0.53 | 0.19 | 5.10 | 1.02 | 2.47 | 0.09 |
| 17 | 0.074 | . ... | 1.34 | 0 13 | 3.27 | 1.73 | 1.42 | 0.09 |
| 19 | Through 0.074 | ....... | 2 89 | 0.24 | 11.07 | 3.36 | 4.71 | 0.06 |
| | Aver. size of particle, mm....... | 13.750 | 9.600 | 5.170 | 0.583 | 5.280 | 2.810 | 11.720 |
| | Total energy units | 204.37 | 382.60 | 527.30 | 1,269.69 | 600.74 | 853.40 | 940.75 |

TABLE 2.—*Screen Tests.*—(*Continued*)

| Ordinal Number | Screen Aperture, Millimeters | 5331 | 1 | 2 | 3 | 4 |
|---|---|---|---|---|---|---|
| −1.0 | 38.100 | | | | | |
| 0 | 26.670 | ...... | ... . | .... .. | 7.81 | |
| 1 | 18.850 | . ... | 1 14 | . ... | 8.78 | |
| 2 | 13.330 | ...... | 7 87 | 0.24 | 25.76 | |
| 3 | 9.423 | .. ... | 18 60 | 1.00 | 26.31 | 1.32 |
| 4 | 6.680 | ...... | 19 05 | 3.36 | 17.42 | 3.88 |
| 5 | 4 699 | 0.07 | 13.22 | 5.18 | 9.19 | 6.23 |
| 6 | 3 327 | 0.38 | 13.92 | 6.37 | 2 21 | 5 14 |
| 7 | 2 362 | 1.44 | 9 21 | 6.91 | 0.68 | 5.48 |
| 8 | 1.651 | 5.28 | 4 81 | 6.75 | 0.48 | 4.13 |
| 9 | 1.168 | 8.12 | 2.75 | 7.00 | 0.24 | 3.06 |
| 10 | 0 833 | 19.48 | 2.21 | 7.90 | 0.15 | 2 93 |
| 11 | 0 589 | 20.70 | 2 00 | 8.73 | 0.14 | 4 07 |
| 12 | 0 417 | 15.98 | 1.41 | 8.94 | 0 11 | 5.13 |
| 13 | 0 295 | 12.25 | 1 16 | 7.64 | 0.12 | 7.14 |
| 14 | 0 208 | 6.89 | 0 78 | 6.10 | 0 11 | 7.21 |
| 15 | 0.147 | 5.02 | 0 59 | 5.75 | 0.11 | 7.68 |
| 16 | 0 104 | 2.12 | 0.52 | 5.21 | 0.12 | 8.45 |
| 17 | 0 074 | 0.96 | 0 23 | 1.56 | 0.07 | 4 99 |
| 19 | Through 0 074 | 1 31 | 0 53 | 11.36 | 0.19 | 23.18 |
| | Aver size of particle, mm | 0.644 | 5 746 | 1.350 | 11.350 | 1.219 |
| | Total energy units ... . | 1,148.81 | 534.39 | 1,125.61 | 294.75 | 1,284.98 |

TABLE 3

| Test No. | Feed Rate, Pounds per Hour | Aver. Size of Feed, Millimeters | Aver. Size of Product, Millimeters | Ratio of Reduction | R.M.E. |
|---|---|---|---|---|---|
| 202 | 1,000 | 24.58 | 0.142 | 173:1 | 9.30 |
| 203 | 2,000 | 24 58 | 8 590 | 2.88:1 | 10.30 |
| 204 | 1,500 | 24.58 | 6.573 | 3.74:1 | 9.13 |
| 213 | 1,500 | 9 90 | 0.288 | 34.2:1 | 9.86 |
| 216 | 3,000 | 9.90 | 0.388 | 25.5 :1 | 17.30 |
| 217 | 6,000 | 9 90 | 0.768 | 12 9 :1 | 29 21 |
| 218 | 12,000 | 9 90 | 2.129 | 4.65:1 | 41.10 |
| 219 | 9,000 | 9 90 | 1.628 | 6.08:1 | 43 50 |

by one of these mills is considered, together with the fact that the power consumption is practically the same whether the mill is loaded lightly or heavily, it should be apparent that it will pay well to expend the small amount of power necessary for handling the pulp in a closed circuit and thereby gain increased efficiency of the mill.

This conclusion is practically in accord with that reached by the usual method of analysis. Taking 0.295 mm. (48-mesh) as a limiting size sought, Table 4 shows the relative number of mills and, therefore, the relative amounts of power necessary to crush quartzite, the screen test of which is

shown in S. T. 39-A, to pass a 0.295-mm. screen at the rate of 12,000 lb. per hour. The same relative figures will, of course, hold for any multiple of this desired capacity.

TABLE 4

| Test No. | Feed Rate, Pounds per Hour | Total Feed Including Returns from 12,000 Lb per Hour Original Feed, Pounds per Hour | Minus 0 295-Mm Material in Feed, Pounds per Hour | Minus 0 295-Mm Material in Product, Pounds per Hour | Minus 0 295-Mm Material Produced, Pounds per Hour | Number of Mills Needed |
|---|---|---|---|---|---|---|
| 213 | 1,500 | 20,000 | 104 | 896 | 792 | 13 0 |
| 216 | 3,000 | 23,920 | 208 | 1,507 | 1,299 | 8 0 |
| 217 | 6,000 | 36,400 | 416 | 1,978 | 1,562 | 6 0 |
| 219 | 9,000 | 45,000 | 624 | 2,382 | 1,758 | 5 0 |
| 218 | 12,000 | 51,600 | 832 | 2,778 | 1,946 | 4 3 |

This table is based on the assumption that the efficiency of reduction is the same on the smaller material returned to the mill as it is on the larger original feed. This is not quite true, but it is nearly enough true for the purposes of this argument.

Fig. 10 presents a fact which goes far toward explaining the irregular performance often met with in machines following a crusher of the ball- or tube-mill type. It will be noted that at the end of the 6¾ hr. operation the divergence between feed and discharge rate at any given minute is as great as at the beginning of the run, despite a careful, regular feed. The rising portions of the curve are accompanied by a progressively coarser product. At the peaks the screen tests show but little crushing. This irregularity in discharge rate and character of product is greater in dry crushing than in wet crushing, but it is also distinctly apparent in wet crushing. In most mill practice the irregularity is smoothed out by crushing in closed circuit, the circuit acting as a balance. Where no such balance occurs through other features of mill design, it will be wise to make special provision if the machines treating the discharge require a close adjustment.

*Effect of Moisture Content*

Fig. 8, summarizing tests 210 to 215 inclusive, Table 1, shows distinctly the effect of moisture on the crushing efficiency and average size of product of the conical ball mill. The true maximum of the efficiency curve probably lies somewhere between 40 and 50 per cent. moisture. The decidedly higher efficiency of wet crushing over dry crushing is confirmed in tests 207 and 208 where the relative mechanical efficiency rises from 9.33 to 11.46 due to the addition of 19 per cent. water, which is decidedly less than the most efficient water quantity. It will be noted, however, on referring to screen tests 40-1, 41A-1, 41B-1, 41C-1, 41D-1, and 41E-1, Table 2, that a progressively finer product is obtained by in-

creasing the amount of water in the feed and that the decreased relative mechanical efficiency in tests 214 and 215 is due to increased power consumption.

Moisture content has an effect on the weight of crushing charge that can be held in a mill. If the mill is charged to the limit when pulp of a given moisture content is being fed, a slight decrease in the moisture content will cause the discharge of a considerable quantity of balls or pebbles, as the case may be. The converse of this statement is, of course, also true.

*Effect of Slope*

The principal factors in mill operation affected by changes in slope are the ball load and the character of the product.

The effect on the ball load is best shown in tests 205 and 206. At the end of test 205 the mill, then level, contained a charge of 4503 lb. consisting of 2455 lb. of 5-in., 1110 lb. of 4-in., 703 lb. of 3-in. and 235 lb. of 1¾-in. balls. At the end of test 206, which started with this load and was continued for several hours, the mill being set at a slope of 0.64-in. per foot, there had been forced out of the mill 55 lb. of 5-in., 588 lb. of 4-in., and all the 3-in. and 1¾-in. balls, leaving a total charge of 5-in. and 4-in. balls weighing but 2922 lb. The ratio of weight of rock in the mill to weight of balls was also reduced. This latter fact considerably lessens cushioning and increases the amount of crushing done by impact as compared to that done by abrasion. The result is reflected in the increased efficiency and more granular product obtained, as noted later.

The change in power required to operate at higher slopes is in no way commensurate with what would be expected from the decrease in ball load (see tests 205 and 206). This fact should be borne in mind in using the formula given for horsepower.

The effect of changes in slope on the relative mechanical efficiency is so small that contradictory results due, no doubt, to unavoidable experimental inaccuracies, are shown. Thus tests 212, 213, 220 and 221 show a point of least efficiency at 1¼-in. slope with higher efficiencies at 2⅝-in. slope and no slope. It is the writer's opinion that the relative mechanical efficiency increases with increase in slope within operating limits, but that the change will in all cases be small. Tests 222 and 223 compared show higher efficiency and finer grinding at the greater slope. The finer grinding in test 223 is due to the higher moisture content of the pulp, rather than to the increase in slope. The progressively coarser grinding with increasing slope in the quartzite series, tests 212, 213, 220 and 221, is typical of the results to be expected in this direction. In dry grinding (see tests 205, 206 and 207), a decided change in the character of the product takes place with change of slope. The material discharged from the mill when grinding with the axis horizontal, contains

a large percentage of − 200-mesh material and a considerable percentage of the coarsest sizes with a decided minimum in the amount of the intermediate sizes. The product of the tilted mill, on the other hand, is more uniform. There is decidedly less coarse material and decidedly less dust, the bulk of the product lying in the intermediate sizes. This difference is undoubtedly due to the difference in the character of the crushing done in the two cases. With the mill horizontal a considerable proportion of the load is rock. This rock acts as a cushion to the falling balls in the mill so that crushing by impact is greatly lessened and crushing by abrasion forms an important part of the work done. In such crushing many of the large particles in the feed are reduced in size but slightly and pass out practically untouched, while such work as is effective produces very fine material. Thus we have the large percentages of very coarse and very fine ingredients in the product. On the other hand, when the mill is tilted the amount of rock that it contains at any time is small in proportion to the crushing load, there is little or no cushioning and the amount of crushing done by impact is large in comparison with that done by abrasion. Under such circumstances a granular product is to be expected.

*Ball Load.*—Varying the weight of the ball load affects the power consumption, fineness of grinding and relative mechanical efficiency. Power consumption increases with increase in the ball load, but the rate of increase in power consumption is not so rapid as the rate of increase of the ball load. Thus in tests 204 and 205 the ball load is increased 12.5 per cent. while the corresponding increase in power is but 3.8 per cent. In tests 222 and 223 an increase in ball load of 51.8 per cent. produces an increase in power consumption of but 4 per cent. It must be noted, however, that in the latter instance the increase in ball load is accompanied by a change in slope and that the mechanical efficiency of the power chain is unquestionably less when the mill is tilted than when it is horizontal. The increase in ball load in test 205 as compared with 204 causes reduction in average size of product, from the same feed, from 6.573 mm. (0.26 in.) to 3.449 mm. (0.14 in.) or 47.5 per cent. This material increase in the fineness of the product, with its corresponding increase in the mechanical value of the pulp, is sufficient to cause an increase of 20.8 per cent. in the relative mechanical efficiency of the machine, notwithstanding the increased power. As noted previously, the writer believes that the apparently contradictory result presented in tests 222 and 223, where the product of the lightly loaded mill is the finer, is due to the increase in percentage of moisture in the latter product and that with the same moisture percentage in both cases a result in agreement with the first case cited would have been obtained. It is, however, the writer's opinion further that the more lightly loaded mill, tilted and with a carefully aligned power chain, should show a higher relative mechanical

efficiency, due to a reduction in power consumed, which would more than compensate for any increase in average size of the product.

*Effect of Difference in Size of Balls.*—A comparison of test 209 with 210 and of test 224 with 225 shows that the larger the average size of ball in the crushing load (up to 5-in. diameter) the smaller the power consumption and the higher the relative mechanical efficiency. It is to be further noted, that a mixture of 5-in. and 4-in. balls crushes finer than a mixture of 5-in., 4-in., and 3-in. balls of equal weight, when the crushing is done dry and the average size of the feed particles is 9.900 mm. (0.39 in.). When the work is done in the presence of water, as in tests 224 and 225, the product when the ball charge is a mixture of 5-in., 4-in., and 3-in. balls is slightly finer (0.509 mm. as against 0.539 mm.) than when the charge consists wholly of 5-in. balls, but in test 224 the moisture percentage was 47.0 per cent. as compared with 40.0 per cent. in test 225. By reference to the section on Effect of Moisture, it will be seen that this result is probably due to the difference in moisture content and that at the same moisture content the 5-in. balls would crush finer than the mixed charge. In any case the difference in fineness in favor of the mixed load is so slight as to fail to justify charging a ball mill working on coarse feed with anything smaller than 5-in. balls. The writer inclines to the belief that the presence of small balls is a hindrance, and that periodical sorting of the charge accompanied by removal of the small balls (less than 3-in. diameter) will increase capacity, decrease power consumption, decrease the average size of the product and materially increase the relative mechanical efficiency.

*Size of Feed.*—Comparison between tests 208 and 223 apparently indicates that the ball mill works more efficiently on a coarse feed (24.58 mm. (0.96 in.) average size) than on a finer feed (10.394 mm. (0.41 in.) average size). In test 223 the percentage of moisture present, 41.7 per cent., is practically that determined most favorable, while in 208, but 19.5 per cent. of water was present in the feed. Notwithstanding this fact' the relative mechanical efficiency in crushing the larger feed is 11.46 as against 10.71 in the case of the finer feed. There is very little difference in the power consumption. This conclusion must, however, be limited by a statement as to the rate of feed, viz., 1500 lb. per hour. The reduction ratio is but 7.02 in the case of the larger feed as against 45.6 for the finer feed. In neither case was the mill fed up to its most efficient capacity. Comparing the results obtained here with those obtained in the rate of feed tests (213 and 216 to 219 inclusive) we may expect that by pushing the capacity in the case of the smaller feed until the reduction ratio is in the neighborhood of 7.0 that the relative mechanical efficiency will rise to about 40, while from the same series of tests it is obvious that lessening the ratio beyond this point in the case of the coarser feed, by increasing the feed rate, would result in lowering the

relative mechanical efficiency. When these facts are taken into consideration, the smaller feed gives most efficient operation.

This conclusion cannot, however, be extended to finer and finer feeds, as is apparent when the pebble mill runs on trap and quartzite, 228 and 229, are compared with the ball mill runs on the same rocks, tests 213 and 223. In the latter tests, with feeds of approximately the same average size, the efficiencies varied by but 7.9 per cent., the trap showing the higher result. In the pebble mill tests the average size of the quartzite feed was 1.173 mm. and of the trap feed 0.380 mm. The corresponding relative mechanical efficiencies were 5.37 and 1.89, a difference of 65 per cent., all of which must be ascribed to the fineness of the feed.

*Length of Cylindrical Section.*—The effect of increasing the length of the cylindrical section in a ball mill is to reduce the relative mechanical efficiency. This is due to the fact that the ball load and power consumption increase with increased length much more rapidly than the fineness of the product increases. Thus, by reference to Table 1 we find that, all conditions being constant other than those noted above, an increase in power consumption amounting to 89 per cent. occurs with an increase in length of cylindrical section from 16 in. to 48 in., the corresponding decrease in average size of product is but 36 per cent., and there is a resulting decrease in relative mechanical efficiency of 40 per cent., the 16-in. mill being taken as the standard of comparison. Therefore, if the desired capacity of a plant is sufficient to justify the installation of more than one mill, additional mills placed in series, each making a relatively small reduction, will be more efficient than an installation which attempts a large reduction ratio in one mill by increasing the length of the cylindrical section.

### *Pebbles vs. Balls*

Test 228 presents the pebble mill working at a reduction ratio of 8.3, which is close to the most economical ratio. Under these conditions the relative mechanical efficiency is 5.37. Test 227 presents a ball mill of the same cylinder length working on a coarser feed but making a reduction of 53.6 to 1. Even under this unfavorable condition the relative mechanical efficiency is 5.88. If the rate of feed is raised and the reduction ratio correspondingly lowered to a point comparable with the pebble mill, we may expect a relative mechanical efficiency much higher. It is obvious, then, that the ball mill is a more efficient crushing machine than the pebble mill. It is also obvious that it will grind as fine as the pebble mill, when the products of the two tests above cited are compared. The ball-mill product is 0.184 mm. average size, produced from a 9.9-mm. feed, while the pebble-mill product is 0.140 mm. average size produced from a feed only 1.173 mm. average size. Given a feed of the same size, the ball-mill product would have been finer.

Records were kept throughout of ball and pebble consumption, but the results were so contradictory, due to the relatively short duration of the runs, that they are not worth presenting.

### *Character of Feed*

When the feed to a ball mill is a rock similar to an average ore, no great difference in efficiency is noticeable as between different kinds. Tests 221 and 222 give a comparison of grinding efficiencies on quartzite and trap of approximately the same average size. The reduction ratios in the two cases are 40.2 and 37.8 respectively, giving products 0.246 and 0.275 mm. average size. The relative mechanical efficiencies are 9.06 in the case of the quartzite feed and 10.20 in the case of the trap-rock feed. When, however, tough, soft materials such as cocoanut shells or sawdust are tested (tests 230 to 234 inclusive) the efficiencies fall off rapidly to figures ranging from 0.48 to 1.67. This means, of course, that a crushing device employing impact chiefly is not suitable for reducing such material.

An interesting and unexpected result is to be noted in tests 235 and 236. The brass ashes treated in these tests consisted of a mixture of unburned coal, coal ash, and a brittle slag containing metal shot. The coal ash and slag ground up with surprising ease, the coal was easily broken to an intermediate size and then seemed to float through on the surface of the load in the mill, while the metal particles were discharged with very little flattening or abrasion. Thus, due to the heterogeneous character of the feed, a higher efficiency was obtained than would be expected where one of the ingredients was so tough. When, however, it was attempted to grind slowly and pulverize metal, the relative mechanical efficiency of the machine fell to figures ranging from 0.04 to 0.71, confirming the comparison made in the first part of this section between rock and such tough materials as sawdust and cocoanut shells.

### *Capacity*

As stated in the introductory part of this paper, it has not been possible to extend the series of tests to gain capacity figures on mills of different diameters. The writer has, however, some figures on the capacity of 6-ft. and 8-ft. ball mills which indicate that with a feed of average ore of 10 mm. average diameter, grinding wet to pass a 20-mesh (0.833-mm.) screen, the capacity will vary as a function of the cube of the nominal diameter of the mill. Approximate capacities for this duty for mills 4.5 ft. diameter and larger may be derived from the formula

$$C = 0.95D^3 - 65$$

where $C$ = the capacity in tons per 24 hr. and $D$ = the nominal diameter of the mill in feet. This formula should, however, be used with caution.

### *Distribution of Crushing Charge*

At the end of one of the runs on the 4½ by 32 mill, the balls were sorted as they were taken from the mill into heaps corresponding to the portion of the mill from which they were removed. For the purpose of this classification the charge was divided by theoretical vertical planes into five sections, as shown in the diagrammatic sketch, Fig. 9. The heaps taken from each of these sections were then sorted into sizes, with the result shown graphically in Fig. 9. It will be seen from this figure that there is a marked segregation of large balls in sections 1 and 2 at the head end of the mill. The segregation is, however, by no means complete, as is shown by the fact that the average size of ball in the mixture in sections 3, 4, and 5 is greater than 4 in.

## Conclusions

1. In crushing average ores the character of the gangue has but little effect on the relative mechanical efficiency of the conical mill.
2. The mill is not suitable for grinding soft, tough materials.
3. The ball mill works more efficiently on material of intermediate (0.5 in. to 0.75 in. average) size than on either a coarser or a finer feed.
4. A greater ratio of reduction in average size of material can be expected with feed of an intermediate size than with a coarse feed.
5. Steel balls are much more efficient crushing media than pebbles.
6. Steel balls will grind as fine or finer than pebbles when working on the same feed.
7. Increase in the weight of the ball load, other conditions remaining constant, increases the ratio of reduction and the relative mechanical efficiency of the mill.
8. The power consumption increases with increase in the weight of the ball load, but this increase in power consumption is not in direct proportion to the increase in load.
9. Power consumption decreases with increase in the average size of the balls composing the crushing load up to an average size of 5 in.
10. A ball charge composed of 5-in. balls makes a greater reduction in size of particle at one passage through the mill than a mixed charge composed of 5-in., 4-in., and 3-in. balls.
11. The relative mechanical efficiency of the ball mill increases with the average size of ball in the crushing charge up to 5 in. average diameter.
12. The relative mechanical efficiency of the mill increases with the rate of feed to the point of overload.

13. Increase in the length of cylindrical section in the conical ball mill increases the reduction ratio at the expense of a marked decrease in the relative mechanical efficiency.

14. Increase in the slope of the mill axis decreases the ball load materially, but the corresponding decrease in power consumption is in no way commensurate.

15. In general, increase in slope tends to produce a more granular product with less very fine and less coarse ingredients than are present in the product of the mill set with the axis horizontal.

16. Increase in slope has but little effect on the relative mechanical efficiency.

17. Other conditions being constant, the relative mechanical efficiency of the mill is a maximum at between 40 and 50 per cent. moisture content in the feed.

18. The relative mechanical efficiency in wet crushing is decidedly greater than in dry crushing.

19. The increase in the percentage of moisture in the feed causes an increase in the reduction ratio.

20. Power consumption increases slightly with increase in the moisture content of the feed.

21. The rate of discharge and the character of the product of the mill fluctuate continually through rather wide limits. This fluctuation is greatest in dry crushing.

22. The conical mill should be operated in closed circuit with a sizing device which will return to it the oversize from its product. In this installation the rate of feed should be raised until the relative mechanical efficiency shows a maximum. When operating as a ball mill, the ratio of length of cylindrical section to diameter should not exceed 0.3. This will be a much more economical installation than one which seeks, by slow feeding or long cylindrical section, to obtain a finished product at one passage through the mill. In wet grinding the moisture content of the feed should be kept about 40 per cent. The slope should be adjusted to mill requirements, but for ordinary concentrating-mill practice should be about 0.4 in. per foot. The ball charge should be the maximum that the mill will hold and should be kept as large in average size as is possible without too great sacrifice of small balls.

## Acknowledgments

The writer is indebted to Percey F. Smith, Professor of Mathematics in the Sheffield Scientific School, for valuable assistance in the development of the formulæ, and to Professor Herbert L. Seward of the department of Mechanical Engineering for help with the nomograms.

## Discussion

John W. Bell,* Montreal, Quebec, Canada (written discussion†) — The test results in Mr. Taggart's paper will, I am sure, be recognized as a notable contribution, and of great assistance in the study of the performance of the Hardinge mill.

I regret, however, being obliged to note that Mr. Taggart still retains such confidence in the Kick-Stadler method of computing the "relative mechanical efficiency" of crushing machines. The Rittinger-Kick graph submitted by Mr. Gates[1] and the tests made at McGill University disclosed precisely the same fundamental defect in the Kick-Stadler theory.

Consequently, I have been obliged to recalculate in terms of Rittinger surface units the results obtained in the 28 rock-crushing tests cited by Mr. Taggart, in order to find out what the relative efficiencies really were. In order to show the large discrepancies between the Stadler and Rittinger R. M. E.'s, the most efficient result disclosed by each method is represented by the number 100, and the R. M. E.'s for the other tests have been recalculated on this basis. The results will be found in Table 1. Personally, I look forward to the time when we shall cease to talk about "relative mechanical efficiency" and merely refer to the "efficiency" of a crusher. All that is required to accomplish this is to agree on a standard method for determining the "crushing constant" of a given rock and a standard method for calculating the efficiency. The figures in the fourth column of Table 1 have been derived by assuming a constant of 2000 for the quartzite crushed in the Yale tests.

The Stadler method sometimes indicates changes in efficiency produced by changes in operating conditions, as I pointed out in a paper describing rock-crushing tests made at McGill University. It is, however, not enough to determine that certain changes increase or decrease efficiency; surely it is equally important to determine the magnitude of these variations. By examination of Mr. Taggart's results, I have been obliged to conclude that the size of feed and amount of reduction greatly influence the Stadler R. M. E. figures, that they are positively misleading.

It should be mentioned that since the majority of the tests have been made at the least efficient feed rates, and since, moreover, it is one of the evidently very important factors affecting efficiency, it is possible that different results might be obtained by a high-tonnage feed series, and that some of the conclusions reached by Mr. Taggart or by me may require revision when this data has been obtained.

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† Received April 28, 1917.

[1] A. O. Gates, *Trans.* (1915), **52**, 898, Fig. 20.

TABLE 1

| Test No | Apparent R. M. E. Stadler | R. M. E. Rittinger | Efficiency Assuming Quartzite Crushing Constant = 2000 S Units |
|---|---|---|---|
| 202 | 21.4 | 56.6 | |
| 203 | 23.7 | 58.5 | |
| 204 | 21.0 | 54.0 | |
| 205 | 25.4 | 63.0 | |
| 206 | 19.9 | 44.1 | |
| 207 | 21.4 | 47.6 | |
| 208 | 26.3 | 55.1 | |
| 209 | 18.6 | 39.4 | 10.5 |
| 210 | 16.7 | 32.8 | 8.7 |
| 211 | 18.8 | 33.8 | 9.0 |
| 212 | 21.4 | 45.3 | 12.0 |
| 213 | 22.6 | 49.4 | 13.1 |
| 214 | 22.2 | 54.0 | 14.3 |
| 215 | 21.6 | 52.2 | 13.9 |
| 216 | 39.8 | 72.2 | 19.2 |
| 217 | 67.1 | 90.6 | 24.0 |
| 218 | 94.5 | 100.0 | 26.5 |
| 219 | 100.0 | 98.6 | 26.2 |
| 220 | 19.4 | 40.2 | 10.7 |
| 221 | 20.8 | 48.5 | 12.9 |
| 222 | 23.5 | 56.0 | |
| 223 | 24.6 | 69.2 | |
| 224 | 22.6 | 40.6 | 10.8 |
| 225 | 24.6 | 42.1 | 11.2 |
| 226 | 16.4 | 41.0 | 10.9 |
| 227 | 13.5 | 34.4 | 9.1 |
| 228 | 12.3 | 59.8 | 15.7 |
| 229 | 4.3 | 24.5 | |

But even more striking than the errors in magnitude of Stadler R. M. E.'s, are the errors they lead to in some of Mr. Taggart's principal conclusions. He says that (p. 141) the R. M.E. of the machine increases with the feed rate up to 108 tons per 24 hr. "beyond which we have an apparent condition of overloading." My conclusion is that the 144-ton feed rate (test No. 218) is the more efficient, and that there is consequently no indication of overloading. As an additional argument in favor of surface rather than energy units, I have plotted the results (see Fig. 1) of tests 213, 216, 217, 218, 219 given in Table 3 of Mr. Taggart's article, with the apparent Stadler R. M. E.'s and Rittinger R. M. E.'s figured on the same basis as in Table 1 of this discussion. That the Stadler R. M. E. should shoot up to a maximum value at 108 tons in a nearly straight line, and then down at 144, does not seem to me to be what one would reasonably expect. The Rittinger curve seems far more rational.

In regard to the result listed in Mr. Taggart's Table 4 (p. 148), I

repeat a protest I have already made against the estimation of efficiency by considering the number of tons of −48-mesh material produced. The impression that is created by this table is, that a small feed will require 13 tube mills to produce 12,000 lb. per hour of −48-mesh material; whereas a high feed will require only four. On the assumptions made, numerically, this may be true, but a very practical consideration in this connection is that the 13-tube-mill plant will produce a product containing 21.6 per cent. of −200 grade and the 4.3-tube-mill·plant product will only contain 4.7 per cent. of −200. Table 2 shows that by the addition of 1.7 tube mills, the amount of −200 could be nearly doubled with a

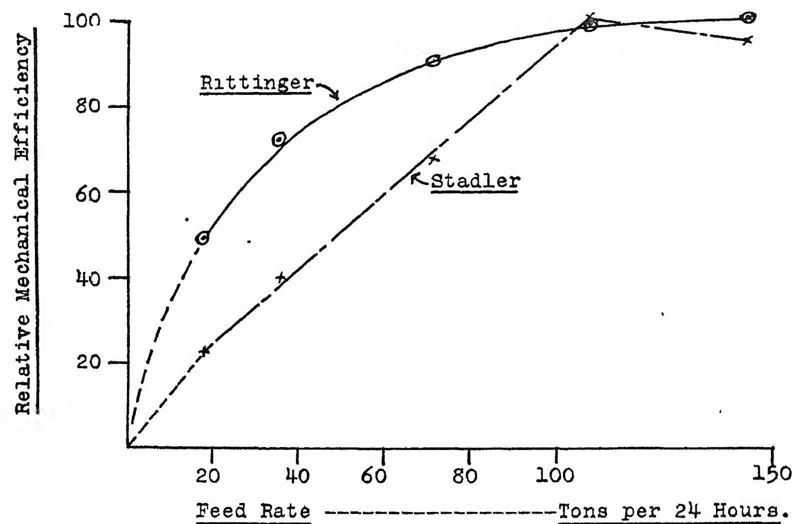


FIG. 1.

drop of only 2½ per cent. in the mechanical efficiency of the mills. The efficiencies are real (assuming the crushing constant to be 2000) in order to eliminate the exaggeration of the effect of tonnage feed created by calculating R. M. E.'s.

TABLE 2

| Test No. | Tons Per 24 Hr | Per Cent −200 in Discharge | Efficiency |
|---|---|---|---|
| 213 | 18 | 21.6 | 13.1 |
| 216 | 36 | 15.2 | 19.2 |
| 217 | 72 | 8.4 | 24.0 |
| 219 | 108 | 5.5 | 26.2 |
| 218 | 144 | 4.7 | 26.5 |

Speaking of his Table 4, Mr. Taggart says: "This table is based on the assumption that the efficiency of reduction is the same on the smaller

material returned to the mill as it is on the larger original feed." When it is considered that the Stadler R. M. E. figures cited by Mr. Taggart for a coarse-feed test and the finest-feed test are as 100 to 4, approximately, it is clear that the assumption is dangerous, and is even dangerous by the Rittinger theory which gives a ratio of 100: 24½.

This naturally brings up the question of whether the oversize product from a ball mill should be returned to the ball mill or passed along to a second grinder (either ball or pebble mill) for final reduction.

On p. 152 Mr. Taggart compares tests 227 and 228 and says: "It is obvious, then, that the ball mill is a more efficient crushing machine than the pebble mill." My conclusion, arrived at by the Rittinger theory, is diametrically opposite, as will be noted by Table 3. That the pebble mill is much more efficient than the ball mill is well shown by the results in the last two columns.

Table 3

| Test No. | | Apparent R. M. E. Stadler | R M E Rittinger | Tons —48-Mesh Material Per Horsepower |
|---|---|---|---|---|
| 227 | Balls | 13.5 | 34.4 | 0.38 |
| 228 | Pebbles | 12.3 | 59.8 | 0.65 |

The fact that there was such a great difference in the feed diameter would render the comparison valueless or nearly so if the Rittinger R. M. E.'s had happened to be nearly equal. But since, in spite of this handicap, the pebble mill is able to demonstrate its great superiority, I am extremely doubtful of the advisability of returning anything but the very coarsest pieces in the oversize to the ball mill circuit, as it seems probable that the regrinding of the finer sizes could be done far more efficiently by a second pebble mill working in a closed circuit.

Mr. Taggart's conclusion "that the true maximum of the efficiency curve lies somewhere between 40 and 50 per cent. moisture" does not seem to be very well supported by his moisture-efficiency diagram, Fig. 8 (p. 139), since it would be, if anything, more reasonable to suppose that the maximum efficiency moisture was either 38½ per cent. or that it was somewhere between 25 and 40 per cent. The Rittinger results in Table 4 show that the maximum efficiency moisture will be found between 52 and 68 per cent. and is probably in the neighborhood of 55 per cent. It is worth noting, however, that the actual gain in efficiency realized by changing from a 25 to a 52 per cent. moisture would only amount to about 2½ per cent. of the power used (see 5th column, Table 1, tests 212–214). This of course applies to the tests cited. The increase might be appreciably greater for a large mill, fed at its maximum efficiency feed rate and feed size.

In regard to the efficiency effect of ball load, slope of mill, size of feed, dry versus wet crushing, etc., I do not think that very positive conclusions can be drawn because of the changes made in the mill adjustments before the required data was obtained. I am inclined to think that the trap crushes so much more easily than the quartzite, as to hardly warrant Mr. Taggart's first conclusion, which is based on tests 221 and 222. If the two rocks were similar, the R. M. E.'s would be the same. The Rittinger R. M. E.'s show an appreciably greater number of surface units produced per horsepower, and if we assume the crushing constant of the quartzite as 2000, the trap constant (assuming that the mechanical efficiency of the mill was the same in each test) would be about 2300. In test 223 an appreciably larger amount of work was done than in test 213 (see Table 4), and it is interesting to note that although the horsepower increased from 17.7 (in quartz test 213) to 20.1 (in the trap test) the R. M. E.'s are respectively 49.4 and 69.2.

If the mechanical efficiency of the mill was the same in these tests the trap constant would be raised to 2800. It should be noted, however, that in tests 221–222, the ball load is 4264 lb. (and the mill is level)

Table 4

| Test No. | Rock Crushed | Ball Load, Pounds | Work Done per Unit, Surface Units. | R. M. E. | |
|---|---|---|---|---|---|
| | | | | Stadler | Rittinger |
| 213 | Quartz | 2,819 | 258 | 22.6 | 49.4 |
| 223 | Trap | 2,811 | 410 | 24.6 | 69.2 |
| 224 | Quartz | 1,406 | 183 | 22.6 | 40.6 |

whereas in tests 213–223 it is only about 2800 lb., with a mill slope of 2⅝ in. It is, of course, possible that the mill has a higher mechanical efficiency when grinding a softer rock, in which case the calculated constant 2800 would be reduced.

The most efficient feed size is a matter of great practical importance. In regard to this, Mr. Taggart says: "The ball mill works more efficiently on material of intermediate (0.5 to 0.75 in average) size than on a coarser or finer feed" (Conclusion No. 3). There can be no question about the inefficiency of a ball mill working on a very fine feed, but I have the liveliest suspicions of the correctness of this statement in regard to the feed coarser than the grade he fixes as most efficient. The coarse-feed tests (202 to 208) have noticeably high R. M. E.'s, but whether due to the coarse feed or the softer trap rock crushed it is difficult to say.

On p. 151, Mr. Taggart explains why he adopts conclusion No. 3. He points out that the Stadler R. M. E.'s. show the coarser-feed test (208) to be more efficient than the finer-feed test No. 223, but he explains

that the probable reason for this is because the reduction ratio is only 7.0 in test 208 as against 45.6 in test 223. Consequently, he says, "we may expect, by pushing the capacity (feed rate) in the case of the smaller (size of) feed until the reduction ratio is in the neighborhood of 7.0, that the relative mechanical efficiency will rise to 40, while from the same series of tests (the feed-rate tests) it is obvious that lessening the ratio beyond this point in the case of the coarser feed, by increasing the feed rate, would result in lowering the relative mechanical efficiency."

This reasoning is not at all obvious to me, because in my conception the ratio of reduction in the coarse test is more nearly 108 than 7, and in the finer feed test I would fix the ratio of reduction to be 14 instead of 46, and since these figures are diametrically opposed to his in direction, the conclusion to be drawn from them, following his own argument, is also reversed, that is, by increasing the coarse-feed rate, until the reduction ratio was reduced from 108 to 14, the coarse-feed R. M. E.'s. would go up by leaps and bounds as shown by the "feed-rate" tests. (See Fig. 1 of this discussion.) It is quite probable that the high R. M. E. in test 223 can be partly accounted for in this way.

The foregoing will make clear my reasons for believing that some of the numbered conclusions in Mr. Taggart's paper should either be reversed, or commented on, as follows:

1. The indications are that the trap crushes more easily than the quartzite, and that the efficiencies are therefore appreciably affected. The effect of small differences in rock constants is lessened by the fact that crushing machines utilize usefully a comparatively small amount of the power they draw.

4. A greater ratio of reduction in average size of material can be expected with coarse feed than with feed of intermediate size.

5. Pebbles working on a fine feed are much more efficient than balls working on a relatively much coarser feed, on account of the large reduction in the power required to lift equal volumes of pebbles (100 lb. per cubic foot) compared with balls weighing 250 lb. per cubic foot. The powers are indicated to be roughly proportional to the weights per cubic foot given. It is to be expected that if the size of feed to the pebble mill was gradually increased, a feed size would ultimately be reached which could be crushed more efficiently by a ball than by a pebble mill. These conclusions are based on tests 227–228.

12. The relative mechanical efficiency of the mill increases to the point of overload, which, however, was not reached in the tests described.

13. I hardly think Mr. Taggart has sufficient data to draw the conclusion he gives.

14. .... "is in no way commensurate." Note.—Probably on account of the inefficiency of the chain drive.

17. The relative mechanical efficiency for the conditions prevailing

in the moisture series of tests, is at a maximum, when the moisture is at or slightly in excess of 52 per cent. of the weight of the pulp.

18. Probably dry crushing is less efficient than wet crushing, but the decrease does not appear to be very large.

22. It seems probable that the oversize from a ball mill could be more efficiently reduced in secondary mills using pebbles.

A. F. Taggart (written discussion*).—The writer wishes to record his appreciation of the careful study bestowed by Mr. Bell on the paper on Tests on the Hardinge Conical Mill and of the labor expended in translating the data therein contained into such shape as to make them comprehensible to those who use the "surface-unit" method of analyzing crushing data.

He wishes further, however, to register emphatic disagreement with the conclusions drawn by Mr. Bell and summarized at the end of his discussion. Most of Mr. Bell's conclusions are so completely at variance with the experience of practical mill men as to make repudiation here superfluous were it not for the fact that they were arrived at by applying a method of calculation ably defended by many writers on crushing data and therefore not to be lightly ignored.

To refer in detail to a few of Mr. Bell's criticisms: (*a*) As to the insufficiency of the data: The writer realized throughout the course of the experiments that rigorous proof of the conclusions drawn demanded more work than it was possible to do under the conditions that obtained. For that reason the data upon which the conclusions were based were fully presented in order that each reader might himself judge of their sufficiency. The writer further corresponded with and talked with several operators of mills before submitting the paper for publication, in order to determine whether or not the conclusions reached differed radically from mill experience, and was pleased to find remarkable agreement.

(*b*) Mr. Bell apparently overlooks the fact that the point of overloading in the operation of any crushing machine marks a sudden change in the phenomena involved. In some machines, such as rolls, stalling occurs; in ball mills there is a practical cessation of grinding, the mill acting as a conveyor only. Bearing this point in mind, the Stadler curve in Mr. Bell's Fig. 1 is more rational than the Rittinger curve, which latter would indicate a broad maximum and a gradual diminution in efficiency as the point of overload is passed.

(*c*) In regard to Table 4, the writer is far from defending the mill method of using "per cent. − 48 mesh" or any other mesh as a measure of crushing efficiency. However, such a means of measurement is used as a guide for practical work by intelligent operators of wide

* Received June 27, 1917.

experience and carries weight for that reason. The near agreement reached by its use with the conclusions of the writer is not the least argument in favor of the Stadler method of measurement.

(*d*) Relation between relative mechanical efficiency and metallurgical treatment: No attempt was made in the original paper to analyze the suitability of any particular product to subsequent mill operations. The requirements of these operations differ with every ore and every process. The effects of changes in the operating conditions on the relative mechanical efficiency of crushing are in no way changed by these other matters. Such analysis can be left to the mill manager.

(*e*) Effect of size of feed on the efficiency of reduction: In discussing the writer's Table 4, Mr. Bell says:

> "When it is considered that the Stadler R. M. E. figures cited by Mr. Taggart for a coarse-feed test and the finest-feed test are as 100 is to 4 approximately, it is clear that the assumption is dangerous, and is even dangerous by the Rittinger theory which gives a ratio of 100 : 24½."

Mr. Bell has apparently compared tests 219 and 229, where practically the only similar conditions are that the tests were performed in the same laboratory with the same percentages of moisture, while for such comparison the only variable should be the size of feed.

(*f*) Pebbles *vs.* balls: It is practically the universal experience, where tests have been run in the mills, that the amount of grinding done in a ball mill per unit of power far exceeds that done in the pebble mill, and that it pays to install the additional power necessary and use balls instead of pebbles. Such a change has been made in many of the mills throughout the country. Mr. Bell notes that the pebble mill produces more "−48-mesh" material per horsepower expended than the ball mill. On the same basis of reasoning, the tube mill is a far more efficient crusher than the gyratory, yet such is not the usual conclusion of mill men. In test 227 the feed was 9.900 mm. average size, in test 228 it was 1.173 mm. Obviously −0.295-mm. material can be produced with a smaller expenditure of power in test 228 than in test 227.

(*g*) The effect of moisture, point 213, Fig. 8, is obviously an accidental maximum, the position of which was determined by the moisture content in that particular test. It will be apparent to anyone accustomed to reading curves that such an accidental maximum might occur at any moisture percentage between 35 and 50. But it will be obvious to the same reader that the maximum of a smooth curve averaging the experimental points will lie between 40 and 50 per cent. and, as a matter of fact, very near 40 per cent. The conclusion drawn by Mr. Bell, that the point of maximum efficiency lies between 52 and 68 per cent. moisture, is utterly at variance with all mill experience where the question of most efficient moisture content has been tried out.

(*h*) Size of feed: The writer is unable to follow Mr. Bell's argument

under this heading, since the definition of ratio of reduction used by Mr. Bell is so widely divergent from the common definition, viz.:

$$\frac{\text{Average size of particle in feed}}{\text{Average size of particle in product}}.$$

The fallacy of his method is proved by mill experience, which has taught operators to feed ball mills with a product in which a large percentage will pass a 1-in. ring, whenever the plant is of sufficient capacity to justify the installation of heavy rolls or disk crushers between the breakers and the ball mill.

(*i*) Conclusions:

1. The writer can see no reason from his data or Mr. Bell's analysis of the same to change his conclusion No. 1. There are unquestionably ores so hard and ores so soft that a comparison of the relative mechanical efficiencies of the conical ball mill working on two ores at the extremes of the list would show a marked difference, but for average ores the writer still believes that the character of the gangue has little effect on the relative mechanical efficiency of the mill.

4. As previously mentioned, Mr. Bell's definition of reduction ratio precludes discussion on this point.

5. This conclusion in the original paper is almost unanimously supported by mill experience.

12. Mr. Bell is working under the disadvantage of not having seen the experiments and not visualizing accurately from the screen tests reported.

13. An operator of mills will have little trouble in agreeing with this conclusion.

14. Chain drive was not used on the mill in the Hammond Laboratory.

17, 18, and 22. The conclusions drawn here by Mr. Bell, using the "surface-unit" method of measurement, are the strongest arguments against the method that the writer has yet seen.

John W. Bell (written discussion*).—Mr. Taggart's reply illuminates a puzzling element in his original paper. Unconsciously, he has allowed practical considerations to influence some of his conclusions. In certain cases, he accepts the direct conclusions indicated by his Stadler efficiencies, and these are proved wrong by *both* the Rittinger and practical method for estimating efficiency. In other cases, a direct conclusion based on his Stadler figures would be so unsatisfactory to him that he is compelled to argue how he could have obtained satisfactory results if he had done something that he did not do; and he bases his final conclusion on the hoped for result. Perhaps the best evidence of his perfect sincerity in arguing as he did is that in one

* Received Aug. 23, 1917.

instance I was just as completely mislead by the plausibility of the argument as he was himself. I failed to note that the very fact that he was compelled to argue a reason for non-acceptance of the Stadler facts, was proof in itself that Stadler's theory had tricked him. The only value my discussion of Mr. Taggart's paper has, will be the proof that Stadler's theory will mislead him, and others, in the future, if they continue to employ it, just as it has misled Mr. Taggart in the present instance. A part of the proof has been submitted and the rest follows.

Let us first consider the facts and the Taggart argument relating to the efficiency effect of "size of feed." The main facts are that Mr. Taggart and Mr. Young made two tests to find out whether a coarse feed (22A) was favorable to an increase or a decrease in efficiency in comparison with an intermediate feed (39A). The efficiency figures were calculated in Stadler energy units. Now if I ask Mr. Taggart—Did the ball mill work more efficiently on the *coarse* feed than on the intermediate feed *in the tests he made* to determine this point?—he is obliged to answer, Yes. My reply is that both the Rittinger and −48-mesh efficiency figures show that the ball mill worked more efficiently on the *intermediate* feed *in the tests he made* (which are the important ones in this discussion) and that consequently his conclusion 3 is supported by the Rittinger and −48-mesh efficiency figures and opposed by his own efficiency figures. Not appreciating that experimenting with a bomb and experimenting with the aid of Stadler's theory are almost equally safe occupations, Mr. Taggart was obliged to argue a reason for avoiding a direct conclusion based on his efficiency figures, and it is a very pretty argument until it is examined closely. The argument is this: that the reason for the lower efficiency indicated by his intermediate-feed test was that there was too much crushing done, or, as Mr. Taggart expresses it, too great a reduction ratio. In short, the very reason that correctly explains why the intermediate-feed test *was* more efficient than the coarse-feed test is the reason Mr. Taggart gives for its being indicated to be less efficient by the Stadler efficiency figures.

The screen analyses of the coarse and intermediate feed discharges afford indirect evidence of the correctness of the Rittinger and − 48-mesh findings, since the coarse-feed discharge contains 17 per cent. of material coarser than 6.7 mm. whereas the intermediate feed contains none. Evidently the coarse feed was too coarse for efficient reduction with 4 and 5-in. balls. On the other hand, if 29 per cent. of fine material was eliminated from the intermediate feed, it seems reasonable to suppose that a much higher efficiency would have been attained.

In my first analysis of Mr. Taggart's data, instinct told me correctly that Mr. Taggart's conclusion No. 3 was not justified by his facts. Misled just as completely as he by an argument with fatal defects, the contradiction between his figures and his conclusion seemed to be

explained by the difference between the diameters he gave and the diameters I found by taking the reciprocals of the mechanical values of feeds and discharges. By completely disregarding my Rittinger data, especially the "work per ton," I unconsciously countered a fallacy with a fallacy and I would not be frank if I failed to acknowledge it. All that was needed to obtain the correct explanation, was to reason it out again and pay attention to my Rittinger facts. Mr. Taggart was quite right in claiming that his ratio of reduction conforms with the usual definition but after he notes how dependable Rittinger's theory is, I hope he will agree that the ratio of reduction is a quantity which is valueless in connection with crushing tests, for the reason that it takes 10 times as much power to reduce $\frac{1}{10}$-in. particles to $\frac{1}{100}$ in. as to reduce 1-in. pieces to $\frac{1}{10}$ in., the ratio of reduction being the same in both cases.

Mr. Taggart has evidently failed to appreciate how seriously his own data indicts the Stadler theory, and this will account for statements he has made which I am confident he will withdraw after an unprejudiced examination of the facts in the case.

First, he says that the results indicated by Rittinger's theory are incompatible with the findings of practical men, and secondly, that the Stadler method agrees with the −48-mesh efficiency method. Evidently, Mr. Taggart neglected to work out the −48-mesh figures because all that is required completely to refute these statements is to submit the figures in the following tables.

The tests in Tables 5, 6 and 7 are arranged in ascending order of efficiency, and one has only to note the orderly ascension of both the Rittinger and −48-mesh efficiency figures in Table 5 and the total lack of relation between their findings and the Stadler findings to realize that the assertions quoted above have no foundation in fact. There is an inconsistency between the Rittinger and −48-mesh figures after passing Test No. 222 in the trap tests, which is worthy of note, but after examining this, my conclusion is that the evidence is in favor of the Rittinger figures, for any practical man will be willing to concede that his method does not give proper credit to a machine for its production of −200-mesh material. Taking this defect into account, the agreement between the Rittinger and −48-mesh figures is astonishing. It is so astonishing that I cannot resist the temptation to ask Mr. Taggart to note that both Rittinger and −48-mesh select pebble mill Test No. 228 to be *the test of highest efficiency* in the low tonnage series of quartz tests, while the Stadler method selects it *to be the test of lowest efficiency* in the same series. He might also note (see Table 6) that −48-mesh agrees with Rittinger that a feed rate of 144 tons is more efficient than a 108-ton feed rate. After considering his argument in this connection I have to admit that I cannot imagine how a ball mill could be fed, "so that there would be a practical *cessation* of crushing, the mill acting as a

TABLE 5.—*Low Tonnage Tests*

| Quartz Test No. | Relative Mechanical Efficiency | | |
|---|---|---|---|
| | Rittinger | −48-Mesh | Stadler |
| 210 | 32.8 | 29.0 | 16.7 |
| 211 | 33.8 | 28.2 | 18 8 |
| 227 | 34 4 | 29.8 | 13.5 |
| 209 | 39.4 | 33.3 | 18 6 |
| 220 | 40.2 | 34.8 | 19 4 |
| 224 | 40.6 | 35.5 | 22 6 |
| 226 | 41.0 | 35.2 | 16.4 |
| 225 | 42.1 | 38.0 | 24 6 |
| 212 | 45.3 | 36.9 | 21.4 |
| 221 | 48.5 | 42.2 | 20.8 |
| 215 | 52.2 | 46.5 | 21.6 |
| 214 | 54 0 | 46 2 | 22.2 |
| 228 | 59 8 | 51 6 | 12 3 |
| Trap Test No | | | |
| 229 | 24 5 | 18.6 | 4 3 |
| 206 | 44.1 | 30 1 | 19.9 |
| 207 | 47 6 | 32 2 | 21.4 |
| 204 | 54.0 | 37 0 | 21.0 |
| 208 | 55.1 | 42.1 | 26.3 |
| 222 | 56 0 | 42.1 | 23.5 |
| 202 | 56 6 | 37 6 | 21.4 |
| 203 | 58 5 | 40 8 | 23.7 |
| 205 | 63 0 | 43.7 | 25.4 |
| 223 | 69.4 | 47.4 | 24.6 |

TABLE 6.—*Feed Rate Series of Tests*

| Test No. | Tons per 24 hr. | Relative Mechanical Efficiency | | |
|---|---|---|---|---|
| | | Rittinger | −48-Mesh | Stadler |
| 213 | 18 | 49.4 | 42 5 | 22.6 |
| 216 | 36 | 72.2 | 66.7 | 39.8 |
| 217 | 72 | 90.6 | 83.8 | 67.1 |
| 219 | 108 | 98.6 | 98.9 | 100.0 |
| 218 | 144 | 100.0 | 100.0 | 94 5 |

conveyor only" unless the feed was inserted by something far more active and powerful than a scoop feeder. At any rate, the evidence in Table 6 satisfies me that instead of the cessation, there was a slight increase in efficiency when the feed rate was increased from 108 to 144 tons. In regard to the relative merits of the Rittinger and Stadler curves, even the Stadler curve fails to afford evidence of a practical cessation of crush-

TABLE 7.—*Moisture Series of Tests*

| Test No | Moisture, Per Cent. | Relative Mechanical Efficiency | | |
|---|---|---|---|---|
| | | Rittinger | −48-Mesh | Stadler |
| 210 | 0.0 | 32.8 | 29.0 | 16.7 |
| 211 | 18.8 | 33.8 | 28.2 | 18.8 |
| 212 | 25.0 | 45.3 | 36.9 | 21.2 |
| 213 | 38.5 | 49.4 | 42.5 | 22.6 |
| 214 | 51.8 | 54.0 | 46.2 | 22.2 |
| 215 | 68.2 | 52.2 | 46.5 | 21.6 |

ing, because after it reaches the 108-ton point, the efficiency decreases gradually. One is compelled to conclude, therefore, that the Stadler critical point indicated is in the nature of a mirage. But perhaps an even more important conclusion to be drawn from Table 6 is the entire absence of merit there is in a method which declares that the R.M. efficiency of a crusher is 23 per cent. when it should be 49 per cent., 40 per cent. when it should be 72 per cent., and 67 per cent. when it should be 91 per cent. Mr. Gates and I both know the reason for this lamentable showing, and we have published it.

In regard to moisture, the results figured by the three methods are given in Table 7. In his reply to my discussion, Mr. Taggart calls for help from the practical men, to support his contention and his curve (suitably modified with a smooth curve) that 40 per cent. is *the* moisture. I am afraid his supplications will fall on deaf ears when practical men find that their own method of figuring efficiency supports the Rittinger conclusion that the high moistures result in highest efficiency. The only difference in the findings is that −48-mesh selects the highest moisture and Rittinger the next to the highest, but both show that there is very little change in efficiency after reaching 52 per cent. moisture. It does not seem to have occurred to Mr. Taggart that the findings of practice are based on tests carried out under conditions quite different from his and that consequently agreement with mill practice is not proof of the correctness of his conclusions. At any rate, Mr. Taggart will appreciate how dangerous it was for him to state a definite conclusion about moisture when he notes that the method he used indicated a change of only 1 per cent. in efficiency in going from 38 up to 68 per cent. moisture and that it indicated a decrease instead of the increase shown by the other two methods.

In his reply, Mr. Taggart claims that he did not analyze the suitability of any product to subsequent mill operations, but he forgot, for the moment, his conclusion No. 22, in which he tells all and sundry to return a ball-mill oversize to a ball mill. This conclusion conforms with his

other conclusions that a ball mill is more efficient than a pebble mill, and that balls are more efficient than pebbles, but all of these are founded on two tests, Nos. 227–228. Here again, he could not reconcile himself to acceptance of a conclusion his efficiency figures pointed to, which was, that for all practical purposes there was but little difference (only 1.2 per cent. in favor of the ball mill) between the efficiency of the ball-mill test and the pebble-mill test, and he is compelled to argue a reason for the trifling difference in favor of the ball mill, and a remedy for rectifying it. Now I think Mr. Taggart will agree with me, that it would be impossible to establish the relative merits of pebble and ball mills unless the several machines were operated under the special operating conditions each required to attain maximum efficiency. Therefore the only question of any importance in connection with Tests 227–228 is whether the ball mill made the execrable showing indicated by Rittinger and −48-mesh efficiency, or was the opposite conclusion indicated by the Stadler method another one of its disconcerting tricks. In this one instance, I cannot give an absolute proof of its error because the operating conditions were so vastly different in the two tests, but a sufficiency of proof will be found in Tables 5, 6 and 7. The force of a conclusion, which Mr. Taggart says is obvious in paragraph (*f*), is lessened almost to the vanishing point when it is remembered that the amount of crushing done by the pebble mill was only 13 per cent. less than the amount done by the ball mill, and was accomplished by the pebble mill with an expenditure of *half the power* drawn by the ball mill. This suggests the possibility that *a too heavy ball load* had something to do with the poor performance of the ball mill although such an explanation is directly opposed by Mr. Taggart's conclusions about ball loads. However, one has only to examine the facts to see that Stadler's theory again failed to supply Mr. Taggart with the evidence required to form correct conclusions about ball loads. The more important facts in this connection will be found in the following table.

TABLE 8

| Test No. | Ball Load | Relative Mechanical Efficiency | | |
|---|---|---|---|---|
| | | Rittinger | −48-Mesh | Stadler |
| 204 | 4006 | 54.0 | 37.1 | 21.0 |
| 205 | 4503 | 63.0 | 43.9 | 25.4 |
| 223 | 2811 | 69.4 | 47.6 | 24 6 |
| 222 | 4264 | 56.0 | 42 1 | 23.5 |
| 224 | 1406 | 40.6 | 30 9 | 22.6 |
| 213 | 2819 | 49.4 | 42 1 | 22.6 |

Mr. Taggart's conclusions about ball loads were founded on Tests 204–205, Tests 223–222, and an argument to explain, as he says, "the apparently contradictory result presented in Tests 223–222." Now, in justice to Mr. Taggart, I must point out that in the sentence following the one quoted, he evidently suspected that the more lightly loaded mill should have been more efficient, although the two sentences seem contradictory. Nevertheless, he concluded that "increase in ball load increases efficiency" and that "the ball load should be the maximum the mill will hold." Here again Mr. Taggart was misled by the colorless indication of his Stadler efficiency figures in Tests 223–222. The Rittinger and –48-mesh figures show in an unmistakable way that the heavy ball load *decreased* the efficiency of the mill in Test 222 and the decrease would have been greater if the power chain (which I inadvertently called the chain drive) effect had been equal in both tests. Evidently there is a ball load of maximum efficiency, depending on the test conditions, above which and below which there is a decrease in efficiency. Probably, therefore, the 6654-lb. ball load in Test 227 was adverse to the ball-mill performance. The unreliability of the Stadler method is again illustrated by Tests 224–213 (Table 8) in which the test conditions permit drawing a conclusion about the effect of ball load. The strict neutrality of the energy unit efficiency figures is belied by the findings of both Rittinger and –48-mesh which declare that doubling the ball load in these tests resulted in a decided increase in efficiency.

In paragraph (*e*) Mr. Taggart has ground for objecting to a 108-ton feed rate test, but there are many others in the series to illustrate my argument, and it is well known that a coarse (but not too coarse) feed is much more favorable to mechanical efficiency than a fine feed.

In regard to the comparative merits of a gyratory and a ball mill, it would not surprise me if a ball mill *was* mechanically more efficient than a gyratory, but nobody could express an opinion about this that would be of value, without first comparing the relative powers with the total number of units of crushing produced by each machine.

Conclusion 4 is correct as originally stated by Mr. Taggart, although, for the reason given, the expression "work per ton" is preferable to the "ratio of reduction." It was conclusion 3 that I wished to enter a protest against, which I now do; not because it is wrong by the correct facts but because it is opposed to the Stadler facts. Just why anyone would continue to support and make use of Stadler's theory after considering the evidence against it, is not at all clear to me. Perhaps the new evidence will be more convincing than the first.

R. B. T. Kiliani, New York, N. Y.—I do not care to discuss Mr. Taggart's paper in the light of theory, as that has been very well done by Prof. Bell, but I should like to criticize some of his conclusions, in the light of actual operating practice at plants all over the country.

1. Mr. Taggart's first conclusion is that in crushing average ore, the character of the gangue has little effect on the efficiency of the mill. This, I believe, is not in accordance with the usual practice, since in crushing a hard ore the capacity will be much reduced below what it would be with a softer ore, while the power consumed will be practically independent of the character of the ore, being proportional only to the load of ore and balls in the mill.

2. Mr. Taggart's second criticism is that the Hardinge mill is not suitable for grinding soft, tough material. In answer to this I might mention that the mill is being used for grinding tough, ductile material, such as metallic aluminum, and also for grinding licorice root.

3. He says that the ball-mill works more efficiently on material of intermediate size, say, ½ to ¾ in., than on either coarser or finer feed. This is true as to coarser feed. For the most efficient work, I believe that a ball mill should be fed with material not coarser than 1½ in.; it will handle material up to 3 and even 4 in., but the reduction from 3 or 4 in. to 1½ in. can be done much more cheaply and efficiently by rolls or disk crushers than in the ball mill.

4. As to Mr. Taggart's fourth conclusion, that a greater ratio of reduction can be expected with feed of an intermediate size than with a coarse feed, I have not enough information to express an opinion.

5. His next conclusion is that steel balls are much more efficient crushing media than pebbles. Steel balls are undoubtedly more efficient for crushing coarse feed. On fine material they are also more efficient as to tons per horsepower crushed to 10-mesh, but on fine material I think it will be found that flint pebbles are cheaper than cast-iron balls, per ton of ore, although there will be a saving in power per ton by using cast iron instead of flint. The increased cost of crushing with cast iron will be due to the higher cost of iron at the present time.

6. Mr. Taggart's sixth conclusion coincides with present practice, that steel balls will grind as fine or finer than pebbles when working on the same feed.

7. It is also true that an increase in weight of the ball load, other conditions remaining constant, increases the ratio of reduction and the relative mechanical efficiency of the mill. However, I believe that there is a certain load which is most efficient, and that this is not the maximum load the mill will hold, filled to the center line, but when loaded up to the trunnion line or about 6 in. below the center line.

8. Mr. Taggart's eighth conclusion, that the power consumption increases with increased weight of ball load, but not in direct proportion, agrees with my observations.

9. He says that power consumption diminishes with increase in the average size of balls, up to an average size of 5 in. We have found, when using smaller balls, that the mill requires more power than with the

same load of large balls; this is probably due to the fact that with small balls it takes a large number to make up the same weight, and therefore more friction is produced when those balls roll over each other.

10. Mr. Taggart concludes that a ball load composed of 5-in. balls performs a greater reduction in size of ore at one passage through the mill than a mixed charge composed of 5-in., 4-in., and 3-in. balls. This does not agree with ordinary practice, because we have found that when we want to crush fine in a ball mill, using only 4- and 5-in. balls, we cannot obtain so great a capacity as when we use a certain number of balls of small diameters. The addition of small balls will usually increase efficiency too, if not too numerous.

11. Mr. Taggart's eleventh conclusion, that the mechanical efficiency of the ball mill increases with the average size of ball in the crushing charge up to 5-in. average diameter, has just been answered.

12. His twelfth conclusion, that the relative mechanical efficiency of the mill increases with the rate of feed, to the point of overload, I believe, is correct.

13. He says that increased length of cylindrical section in the conical ball mill increases the reduction ratio, but at the expense of a marked diminution in mechanical efficiency. That larger ratio of reduction is not very pronounced, although it is distinguishable, although in certain cases that increase in efficiency can be taken care of where large capacity per foot of floor space is desirable, and then it may be advantageous to use a mill of larger diameter. However, I do not believe it is good practice, especially when grinding in closed circuit with a classifier, to use a mill having too small a cylindrical section. Better results are obtained with larger diameter and shorter cylinder. That has been proved by some data I obtained recently. One pebble mill of 8-ft. diameter, operated by a 75-hp. motor, was lagged down to a diameter of 6 ft., while another was reduced to 7 ft. diameter; better results were obtained with the 7-ft. than with the 6-ft. mill.

Mr. Taggart's fourteenth, fifteenth and sixteenth conclusions, regarding the slope of the mill, seem to be borne out by present practice.

17. His next conclusion is that, other conditions being constant, the relative mechanical efficiency of the mill is a maximum when the feed contains between 40 and 50 per cent. of water. Professor Bell claims that 58 per cent. water gives better results. In actual mill practice, that will depend on the character of the ore, since a very dirty ore will require much more water than a granular ore not containing much natural colloidal slime. I know cases where it has been necessary to run the mill with 30 per cent. solids; if the pulp were thicker, no crushing would be accomplished. At another plant they are grinding with 75 per cent. solids and getting very satisfactory results, probably due entirely to the character of the ore. As a general rule, to obtain the best results,

I believe that the pulp should be as thick as possible, as is the usual practice with cylindrical tube-mills.

Mr. Taggart's eighteenth conclusion, that the relative mechanical efficiency in wet crushing is decidedly higher than in dry crushing, I believe has been thoroughly proved. His nineteenth and twentieth conclusions seem to be satisfactory. As to his twenty-first conclusion, I am inclined to doubt it, but I have not sufficient information on which to base definite opinion.

22. His last conclusion is that the conical mill should be operated in closed circuit with a sizing device which will return the oversize to the mill. Apparently better results are obtained by crushing in two stages than in one stage. It is perfectly possible to crush in one stage, and in a small plant this is the proper thing to do, owing to the higher initial cost of a two-stage plant; but in a large mill, where sufficient machinery can be installed, two-stage or even three-stage crushing is considerably more efficient than one-stage crushing.

Mr. Taggart says, "When operating as a ball mill, the ratio of length of cylindrical section to diameter should not exceed 0.3. This will be a much more economical installation than one which seeks, by slow feeding or long cylindrical section, to obtain a finished product at one passage through the mill." For fine crushing, the mill should always be operated in closed circuit, by returning the oversize to the mill itself, so long as the ratio of reduction is not too great; that is, not more than, say, from 8 mesh to 48 or 65 mesh, but not from 1 or 1½ in. to 65 mesh in one stage."

Lastly, Mr. Taggart says, "The slope should be adjusted to mill requirements, but for ordinary concentrating mill practice should be about 0.4 in. per foot." If the inclination of the mill axis is too great it will diminish the ball load unnecessarily. The inclination should not be over 0.2 in. per foot, and when the mill is operated in closed circuit with a classifier, I believe it should be set level. "The ball charge should be the maximum that the mill will hold and should be kept as large in average size as is possible without too great sacrifice of small balls." The ball charge should not be all that the mill will hold, but should be somewhat less than that; neither should it be kept at as large an average size as possible, because by so doing the mill will naturally get all the larger sizes. However, if too many small balls are present, they will probably interfere with the crushing.

Arthur F. Taggart (written discussion*).—The numbers in the following reply to Mr. Kiliani's discussion refer to correspondingly numbered conclusions in the original article.

(1) This conclusion is based on comparative tests with a hard, tough, homogeneous trap, and a rather soft, micaceous quartzite. These two

* Received Dec. 3, 1917.

rocks may be considered fairly representative of the two extremes of average ores. In crushing these rocks, under similar conditions, there was very little difference in the relative mechanical efficiencies of the mill. In coarse crushing, these rocks would have shown considerable difference because the quartzite would break easily along the planes where sericitization had taken place. But when crushing is carried to the point where the quartz or other component minerals of the rock have to be pulverized, and such is always the case in ball- or pebble-mill grinding, the difference disappears. Unless the ore is exceptionally hard and tough, or exceptionally soft and friable (in neither of which cases could it be called an average ore), the relative mechanical efficiency of a mill crushing different rocks under similar conditions will be approximately a constant.

(2) Mr. Kiliani does not state in his criticism of this conclusion what efficiencies are being obtained in the mills he mentioned. The conclusion stated in the original paper was based on performances of the 4½-ft. mill grinding sawdust, cocoanut shells, and metallic copper. Relative mechanical efficiencies ranged, as stated, from 0.04 to 1.67 with the mill pushed to maximum capacity. These figures are to be compared with the efficiency figure 43.50, test 219. The conclusion is, of course, obvious.

(3) The statement of size in conclusion (3) in the original paper is 0.5 in. to 0.75 in. *average*, which covers the range in average size of the product of any coarse breaker delivering material to pass a 1.5-in. ring. Mr. Kiliani apparently agrees with this conclusion. It should be borne in mind, however, that the work on which this conclusion is based was done in a 4½-ft. mill. This size is the lower limit, or, if anything, somewhat below the lower limit in size for ball mills in ore-milling plants. Mr. Kiliani speaks of a ball mill handling 3-in. or even 4-in. material. The 4½-ft. mill will not handle this size at all. The diameter of the mill is not great enough to give sufficient fall to the balls to break such large lumps. It should be expected, therefore, that the larger sizes in the tests presented in this paper would be more satisfactorily reduced in the 6 and 8-ft. mills. These remarks apply also to conclusion (4).

(5) This conclusion was based on crushing efficiencies without regard to the consumption or cost of crushing media. From such a standpoint, it is apparent that Mr. Kiliani's experience agrees. The economic efficiency would obviously vary as between metal balls and flint pebbles, both with locality and with metal and pebble prices. No general statement can, therefore, be made on this score.

(7) The data presented will not bear out Mr. Kiliani's contention that there is a point of maximum efficiency in the ball load below the point of maximum charge which the mill will hold. Nor does such a conclusion seem reasonable. If the assumption is made, and it seems

to the writer a proper one, that crushing is done by all the balls in the mill, then any increase in the ball load which is not accompanied by an increase in the power consumption should add to the efficiency of the operation. This is the case when loading is carried beyond the horizontal axis of the mill (See curves, p. 131).

(10, 11) Mr. Kiliani's facts agree with these conclusions, although he does not draw the same conclusions from them. Any attempt to crush fine in a ball mill (when the term "ball mill" is used to describe a mill taking feed of 1½-in. maximum, or greater), will result in reduction of capacity and in a corresponding reduction in relative mechanical efficiency.

(13) The writer has found that the increase in reduction ratio with a 4½-ft. by 48-in. mill as compared with a 4½-ft. by 16-in. mill is marked, as is also the accompanying decrease in relative mechanical efficiency. The stenographic transcript of this part of Mr. Kiliani's comment is confused, but he is apparently discussing increase in diameter rather than increase in length of cylindrical section.

(17) The only exception which Mr. Kiliani cites to this conclusion is that of a very dirty ore, by which statement it is presumed that he means a clayey ore. Such material is, of course, entirely different from any of that tested in the work leading up to the present paper. Subsequent experience confirms the original conclusion that on average ores the relative mechanical efficiency will be at a maximum with a feed containing 40 to 50 per cent. moisture, and in the usual case the maximum point lies nearer to 40 than to 50 per cent.

(21) This conclusion is confirmed by a considerable number of observations other than those cited in the paper. In fact, no one characteristic has been so invariable in the writer's experience with the conical ball and pebble mills as this one of variation in the tonnage and character of discharge under conditions of constant feed. No tonnage measurements or sizing tests made on a single sample of the discharge should be depended upon as being representative of the average performance of either mill.

(22) Mr. Kiliani questions the statement in regard to the best slope for a mill for ordinary concentrating-mill practice. The advantage of the larger slope is confirmed by the rather remarkable results obtained by cylindrical ball mills using some such means as a sand elevator in the discharge end to aid the egress of fine material. The result of such continuous removal of fine material is to cause the pulp load in the mill to be small, thus doing away with all cushioning and making every blow of the balls effective in crushing ore. The same result can be obtained in the Hardinge mill by giving the mill a very decided slope toward the discharge end. If it is aimed to make a large reduction in the ball mill at one passage, this can be done at small capacity by increasing the

length of the cylindrical section and setting the mill level, but where more than one-stage crushing is practised, and high relative mechanical efficiencies are desired, they can be obtained only by this method.

The criticism of the statement in this conclusion that "the ball charge should be the maximum that the mill will hold and should be kept as large in average size as is possible without too great sacrifice of small balls" is commented upon in paragraphs numbered 7, 10 and 11 above.

JOHN W. BELL (written discussion*).—In his discussion of Mr. Taggart's paper, Mr. Kiliani assigns to me the claim "that 58 per cent. moisture gives better results." I have not claimed, and have no intention of claiming, that 58 per cent. moisture will give better results in practice than the lower percentages of moisture which large-scale tests have shown to be preferable. I do claim, and have submitted the proof, that of the six tests made by Mr. Taggart to determine the effect of moisture on efficiency, 52 per cent. was decidedly more efficient than 38½ per cent. moisture.

I have never had the least doubt that the relative mechanical efficiency of a ball or pebble mill increases to the point of overload. Practical operators offered convincing evidence on this point long ago. But what I do claim is that Mr. Taggart erroneously concluded that he had reached and passed the point of overload, and that the error resulted by his adherence to a theory which falls down hopelessly when it is tested practically, that is, by experiment.

At McGill we have made a number of tests on a trap rock and on a quartz gangue. They are "average" rocks and yet 1 h.p. expended on the quartz will produce 1½ times as many units of crushing as are produced when the same power is expended on the trap. These tests were fine-crushing tests.

Mr. Kiliani says that he does not care to discuss Mr. Taggart's paper in the light of theory, because I have done that. I do not accept his kind impeachment because I think the only practical way to test Taggart's conclusions is to examine them carefully in the light of the facts and figures which have been offered in support of them and against them.

I am sure Mr. Kiliani will agree that it would be desirable to be sure that the Stadler-Kick method of estimating efficiency is so defective as to preclude drawing correct conclusions from its findings. Of a number of proofs of this, perhaps the one which will enable Mr. Kiliani and others to make a quick decision is the fact that in a series of 13 tests, the one test which the Stadler-Kick method indicates as giving the

* Received Feb. 8, 1918.

lowest efficiency is found, both by the Rittinger and the $-48$ mesh method, to give the highest efficiency in the whole series.

How to measure and how not to measure the efficiency of a crusher is a question which should be of interest to practical men. The facts and figures submitted by and in connection with Mr. Taggart's paper are very illuminating in their bearing on this question. It would seem to be in the interest of science and practice to discuss this aspect of Mr. Taggart's paper thoroughly.

# The Enrichment and Segregation of Mill Tailings for Future Treatment*

BY F. E. MARCY,† B. S., SALT LAKE CITY, UTAH

(St Louis Meeting, October, 1917)

It is not my purpose to write a lengthy article or to attempt the solution of the problem I am presenting, but to call attention to what I believe an important issue, hoping that it may arouse in some one, or in some company, a sense of responsibility. I am referring particularly to modern up-to-date milling plants discharging tailings which under existing conditions cannot be further treated or from which a recovery cannot be made at a profit, but where a quantitative and qualitative analysis shows that a portion of these tailings might profitably be treated at some future time when our metallurgical knowledge has advanced, when we have cheaper power, chemicals, labor or other advantageous conditions.

## Early Methods

In our early milling methods very little attention was given to the care of tailings. It was thought that if the mill could be located by a stream or a river where the final disposal of the tailings could be forever lost sight of, it was highly desirable. We can now see and estimate what was lost in these rich tailings, not recoverable from the streams and the great rivers. Fortunately for many, the depositing of tailings in these streams and rivers was objected to by the farmers and finally prohibited by the State Legislatures.

In looking back over our past milling methods, it seems that the first endeavor was to make the maximum amount of money for a particular period, or year, regardless of the future, the depletion of the mine, or the conservation of the total mineral content. The dumping and sluicing of 3000 tons per day, for a period of many years, of a copper tailing carrying 12 lb. of recoverable copper together with its portion of precious metals, the thousands of tons of valuable metal contained in the tailings that have been carried away by Silver Bow Creek, the streams of the Coeur d'Alene, the Flat River and Colorado districts, the enormous quantities

* Originally presented at a meeting of the Utah Section, on May 23, 1917.
† Consulting Engineer.

of the fine slime tailings that were dumped into Torch and Linden Lakes, illustrate such extravagances.

Companies that have been compelled to impound their tailings for the sake of recovering their water are the ones where the best metallurgical work has been done. Where water has been cheap and abundant, too much has been used, which leads to dilution and poor concentrating work. Those milling companies that seemingly were not so favorably located, or the companies that were forced to discontinue the pollution of the streams, have found themselves in the possession of tailings that are of great value, due to the new methods of concentration and of recovery; of these, flotation has taken the foremost position. I am advised that the Canadian Government has taken the stand that the mineral wealth of the country belongs to the country, and the development of it is a privilege granted by the Crown; that the mineral resources of the Dominion cannot be squandered by loose and extravagant methods of extraction. We now have in this country definite laws preventing the scattering of waste products from mills, with the result that today millions of dollars are being recovered from these tailing dumps at a comparatively small expense.

The problem of future treatment of our present mill tailings is a much more difficult one for the future metallurgist and mill man on account of the recent great advancement in our milling methods; yet relatively the valuable content that still remains may have proportionately greater value as our mines become exhausted and our great mineral deposits have become definitely known and their value realized.

I do not share the view of those who consider that flotation is in its infancy, any more than the electrical industry is in its infancy. If so, they are infants of great proportion. In both industries we are fast approaching the maximum efficiencies. Yet the metallurgist will often find residual values from these modern concentrators and if they are not isolated by enrichment and segregation they will be forever lost on account of admixture with valueless material.

## An Illustration of Segregation

I remember that while I was employed by the Anaconda Copper Mining Co., in 1901, this company made arrangements to segregate the fine slimes from the sands, supplementing the arrangement which their new plant afforded in saving all of the tailings instead of allowing a portion of them to go down Warm Springs Creek. It was realized, in starting this new concentrator, that these slimes were of great value on account of the copper content and, consequently, ponds were made, 300 ft. in width, 600 ft. in length and approximately 14 ft. in depth, to settle these values and decant the water. These ponds were operated intermittently—as soon as the ponds became loaded they were drained and then excavated by a drag line and bucket. The discharge from the bucket was piled along the

side of the pond. Part of this slime was made into blast-furnace bricks and smelted, the remaining portion being stored for future treatment.

A more detailed statement of the character of this particular material is found in an excellent paper by Ralph Hayden on The Concentration of Slimes at Anaconda.[1] The effect of saving these slimes for the last 16 years at the Washoe Reduction Plant has resulted already in the recovery of millions of pounds of copper with many millions more yet unrecovered. The present flotation slime plant at Anaconda which is working on these particular slimes recovers more than 1,000,000 lb. of copper per month, which is produced at a cost probably not exceeding 5c. per pound. The Chino Copper Co. segregated the tailings from its mill at Hurley and now has a retreatment plant handling 1000 tons per day with further extension under construction. There are other companies that have segregated to some extent.

There are millions of tons of sulphide tailings that are now becoming slowly oxidized, and cannot be treated without installing expensive grinding machinery plus the flotation equipment; if the slime which carries the high values had been segregated, millions of pounds of copper could easily be recovered by the simple flotation methods of today, as illustrated by the Anaconda and Chino practice. While the sands, which are of lower value, would oxidize faster without the slimes and become available for leaching where coarse material is desirable.

The illustrations that I have given as to our past losses are in no sense given for the purpose of criticising our managers and millmen, for their achievements are all a matter of record that will go down in our metallurgical history; losses and wastes go hand in hand with new ventures and a new country. The thought I desire to set forth is our future responsibility in these matters.

## Kind of Tailings That May Be Segregated

There are often concentrators in operation where some portion of their middlings cannot be brought to grade or a marketable product made. These should be stored separately and not mixed with the general tailings. We have not been able to recover the oxides and carbonates that are mixed with the sulphide tailings. J. M. Callow's filming experiments, which he has so carefully described,[2] point out what may be expected. There are millions of tons of tailings, however, of which the oxide and carbonate contents are less than those of the tailings that are thrown away by the present filming process, and it would seem that enrichment for future treatment should be made if possible. I think it is generally agreed that Mr. Callow is most familiar with the subject of the recovery of complex ores, and he has, at my request, made the following statement

[1] *Trans.* (1913), 46, 239.

[2] *Trans.* (1917), 56, 676.

of his views as to what we may expect as to the probable recovery of such values.

"In the treatment of complex ores, such as those carrying lead, zinc and iron, and in copper ores containing both sulphides and oxides of this mineral, our present methods of recovery are still in embryo, and such tailings must inevitably carry an appreciable portion of the original contents of the ore, in the complex zinc, lead and iron ores especially. Sixty and 70 per cent. zinc recovery is often considered passable work, the rest of the zinc being either entangled or in some other way combined with the iron and lead which future inventions will no doubt show us how to recover. In the copper ores, flotation has added greatly to our past recoveries, but only in the recovery of sulphides; the oxides are still an unavailable asset, which in due time will undoubtedly be recovered by improved methods of extraction.

"In laying out a system for impounding tailings, the plan hitherto has been to impound them without any discrimination as to their values. In some ores it has been found that the principal values lost lie with the sands, and in others, with the slimes; so that a segregation might well be practiced."

As an illustration, let us assume the accumulative analyses of the tailings from one of the great concentrating mills where the process consists of tabling and flotation. The values given are in the form of copper contained in sulphides and carbonates. We will say that an extraction of 94 per cent. is made on the sulphides and 20 per cent. on the carbonates, making a total extraction of 82 per cent. The total copper remaining in the various screen sizes is given in Table 1.

TABLE 1.—*A General Tailing*

| Mesh | Per Cent Ind. Wgt. | Per Cent. Cum. Wgt | Sulphide Cu Content | Oxide Cu Content | Total Cu Content |
|---|---|---|---|---|---|
| + 48 | 1.6 | 1.6 | 0 009 | 0.002 | 0 011 |
| + 65 | 16 2 | 17.8 | 0.030 | 0 021 | 0.051 |
| +100 | 11.2 | 29.0 | 0.016 | 0.012 | 0 028 |
| +150 | 11.0 | 40.1 | 0 019 | 0 016 | 0 035 |
| +200 | 8 0 | 48 2 | 0.013 | 0.013 | 0 026 |
| −200 | 52 0 | .... | 0.051 | 0 188 | 0.239 |
| | 100 0 | | 0.138 | 0 252 | 0.390 |

On such tailings, it is possible with a Dorr classifier to separate the sands from the slime; the slime tailings may be divided again into two parts by deflocculating and decanting, making a fine crystalline mineral and a decanted slime. It is my belief that the decanted material will carry an amount of oxide copper two or three times that of the general mill tailings. While it may not be possible with the present means to make a satisfactory and marketable concentrate, it would appear desirable to store separately rather than to mix with the impoverished tailings which might carry only 0.15 per cent. in copper, and hold such segregated material until metallurgy has advanced or the market is in condition to profitably treat it. I am not offering this as a solution

to the problem of segregation of the tailings, but to point out what some preliminary experiments seem to show.

To retreat 15 or 20 millions of tons of unsegregated tailings in which a large portion carries only 0.12 to 0.15 per cent., is no easy problem for the future metallurgist.

There are many milling plants that treat complex ores which carry lead, zinc and copper, and it is not possible to make a satisfactory extraction from all of these values. In such plants, the extent to which enrichment and segregation of tailings can be carried on is limited only to the extent in which the expenditure seems desirable in the way of a future investment. Whether there ever will be a better opportunity for segregation, or greater ease of separation of these values, and whether it can be done at a lower cost than when the tailings were produced, remains to be seen. Fresh and loosely combined material is always the most desirable to treat in all milling processes.

## Storing and Importance of Milling Site

In general, tailings that contain probable recoverable values should not be allowed to accumulate over large areas or in horizontal layers. Tailings from our first work in concentration generally contain the greatest values. Such tailings should be stored, if possible, so that those that are richest and most easily recoverable can be worked first, thus avoiding handling a large class of inert material when retreatment is commenced. It is certainly not desirable to pile the present low grade of tailings coming from mills where 90 to 95 per cent. recovery is made upon tailings from which only a 65 per cent. recovery was made.

I believe the segregation of tailings can be advantageously accomplished in many instances so that the weathering action will oxidize them for the ordinary leaching. The piling of tailings so that the maximum oxidation can take place, the elimination of all colloidal and talcy material, would seem desirable, and a greater total recovery could be made from leaching and precipitation with iron.

The excellent results obtained at Anaconda in making building brick from segregated tailings is a subject worthy of consideration. It may surprise many to know the extent to which brick making may be successfully carried on to meet immediate market conditions, and the extent to which these brick may be shipped profitably to other markets.

## The Cost of Segregation

Segregation in general can be carried on, I believe, at a low cost, particularly at the time of the initial milling. The operation of tables, Callow tanks, Dorr thickeners, Cole drag-belt separators, excavating machinery and conveying equipment can be done for a very small amount per ton. There are many illustrations of the actual cost of stacking where

mills are located on a flat site, as in northern Michigan and the zinc and lead districts of Missouri.

As to the charge for this work of segregation, it seems it is not a proper milling cost. For the production of this material credit might be given to extinguishment of ore reserves or an entirely separate account be kept. In my opinion, a most effective suggestion has been made by C. W. Van Law, who says: "To my mind this is not at all a charge to be carried against current operations, but is properly chargeable to capital as a deferred asset, just as in the case of the ore that is broken and carried in excess of requirements in shrinkage stope operations on a large scale. The cost of carrying the tailings away from the mill to dispose of them, so far as is necessary merely to enable continuous mill operation, is a proper current operating charge, but anything done in excess of this, where there is reasonable expectation of later obtaining more from the tailings than is at present possible, should be, in my judgment, carried on the books of capital account, their aggregate representing the value of the stored accumulation as an asset at exact cost of producing. Later, when these deposits are reworked, this asset should be scaled down, ton for ton as removed, corresponding reductions in the book asset being made at the same rate as they are created, to the entire extinction of the capital charge, such amortized asset being replaced by the aggregate profits realized from their reworking. I believe this to be an entirely logical and legitimate method of handling this matter, and it is what I should apply in my own case."

This is one of the problems of conservation of our resources and is analogous to many of the problems that have been so recently brought to our attention. The Government no longer allows the careless lumberman to set fire to the brush after he has removed the marketable timber, thus destroying the young trees and leaving the soil exposed and subject to erosion. The importance of a qualitative and quantitative study of these remaining values in tailings and the removal or relative enrichment is a subject for consideration and careful investigation.

If from our up-to-date milling plants millions of tons of copper-bearing mineral are allowed to mix with worthless material, which when so mixed can never profitably be reworked, and if it is possible to segregate a considerable portion of this copper-bearing mineral so that it can profitably be retreated, then the responsibility of trying to save this large quantity of copper must rest with the administration and the board of directors of such companies. It is not sufficient to turn this problem over to the operating mechanical superintendent who may lack metallurgical training, or operating metallurgists who likewise may be handicapped. Therefore, can we not suggest and point out the desirability of segregating and storing such tailings in a fashion that will best facilitate their future working and the saving of valuable metal?

# A Study of the Microstructure of Some Clays in Relation to Their Period of Firing

BY H. RIES* AND Y. OINOUYE†

(St. Louis Meeting, October, 1917)

## Introduction

Of the several interesting physical properties of clay which have claimed the attention of investigators in recent years, none is more important than the behavior of the material when heated to temperatures above those of dehydration.

Some of the problems to be solved are:

1. The mineralogical changes that take place in firing.

2. The relation between temperature and change in constitution, shrinkage, porosity, etc.

3. The importance of the time factor, as duration of firing, either in reaching a given temperature, or of exposure to a constant temperature.

### *Mineralogical Changes*

As a result of fusion, reactions take place between the constituents of a clay, which result in a reorganization of the mass, and the formation of new compounds. Of the new minerals developed, sillimanite is the one usually recorded, and has been referred to by a number of observers.[1] There appears to be a difference of opinion, however, regarding the temperature at which it is formed. Thus Klein[2] notes that sillimanite forms only above 1300° C., while Mellor claims that it can develop below 1200° C., the difference in results being conditioned by the time factor. In Klein's experiments the test pieces were fired in 12 hr., while in Mellor's, the temperature rose from 800° to 1180° in 24 hr., and occupied the same time in cooling back to 800°.

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[1] Vernadsky: *Bulletin de la Société Française de Minéralogie* (1890), **13**, 256; Mellor: *Journal of the Society of Chemical Industry* (1907), **26**, 375; Klein: *U. S. Bureau of Standards, Technologic Paper No.* 80 (1916), 36; and others.

[2] *Transactions of the English Ceramic Society* (1916–17), **16**.

Mellor and Holdcroft[3] state that when clays are fired at 600°, the molecule probably breaks down in free alumina, and free silica, while ferruginous clays also form $Fe_2O_3$. Hussak[4] found that low-burned porcelains show all the constituents doubly refracting and clearly distinguishable from each other, but that higher heating gave an amorphous groundmass, with a few large quartz grains, and needles that were apparently sillimanite.

Klein found that by heating English kaolin for 5 hr. at 600° C., the kaolinite became entirely isotropic, but that the index of refraction (1.55) was only slightly altered. Further, he says that no profound change was noted in the optical properties until 1200° C. was reached, at which point there began incipient dissociation of the kaolinite into two isotropic substances, with complete dissociation at 1400° C, and the development of prismatic grains between 1400° C. and 1450° C.

There is thus a difference of 600° C. in the dissociation point of kaolinite as expressed by Mellor and Klein and there may also be some question as to whether the change from an anisotropic to an isotropic condition does not indicate a change in the mineral constitution.

Another possible change referred to in recent papers is that of quartz to tridymite and cristobalite, but on this point the results obtained by different observers vary, and may be due to the influence of the time factor. McDowell[5] found that upon repeatedly burning silica brick the quartz is transformed to tridymite and cristobalite.

Where we are dealing with impure clays, changes, due to fluxing action, will undoubtedly begin at a much lower temperature than they do in the purer clays like kaolin.

### *Temperature and Time Factors*

It is commonly recognized that as the temperature increases, shrinkage and density increase up to a certain point, beyond which they diminish, but that the time factor is an important one. In other words, certain changes, produced by heating a clay to a given temperature in a given time, may be reached by taking a longer time to heat the same clay to a lower temperature, or by holding it at the latter for a considerable period. This is thought to explain the difference of view regarding the formation of sillimanite expressed by Mellor and Klein, although the latter says that repeated burning of a clay at the same temperature does not produce much change in its constitution.

A criticism sometimes made of laboratory tests on clay is that they lack the long exposure to a slowly rising temperature, obtained in a

[3] *Transactions of the English Ceramic Society* (1911), **9**, 91.
[4] *Sprechsaal* (1889), **153**.
[5] *Trans.* (1917), **57**, 3.

regular factory kiln. While this is true, it is also true that the test pieces used in the laboratory may be smaller than those fired in a large kiln, and require less time to heat through.

The tests in the present paper contain some data on this subject.

## Scope of Present Paper

Most of the petrographic work done on burned clays has been carried out on high-grade clays which contain relatively small amounts of fluxing impurities, and, moreover, the studies have been confined chiefly to results obtained at temperatures of 1200° C. or over.

The present paper deals with some clays that contain notable amounts of fusible impurities,[6] and were fired at temperatures not exceeding 1150° C.

## Method of Procedure

In the present investigation, which is to be regarded as a preliminary one, the clays tested were molded in 1-in. (25.4-mm.) cubes, and 3-in. (76.2-mm.) bars. One set was heated to 1000° C. in 1½ hr. (referred to on subsequent pages as the preheating period), and held there for 10 hr.; a second set was heated to 1150° C. in 3 hr., and held at that temperature for 10 hr.

Samples were drawn at the end of the preheating period, and at hourly intervals thereafter, the test pieces after removal from the furnace being buried in hot sand, so as to cause slow cooling.

The pieces so fired were then tested for fire shrinkage, absorption, porosity. Thin sections were also cut for petrographic study.

The tensile strength of both the air-dried and burned clays was also tested.

### *Clays Selected*

The clays selected for investigation were the following:

1. Brown residual clay from Ordovician shale, occurring at Webster, Botetourt County, Va.

2. Grayish-white residual clay, derived presumably from Cambrian shale, occurring near Lofton, Augusta County, Va.

*Brown Clay.*—The Webster clay is soft, smooth, yellowish-brown when wet, and contains small fragments of unweathered shale. It grinds easily, and mixes with water to a fairly plastic mass. Under the microscope it shows minute flakes of hydromica[7] and kaolinite, as well as small particles of iron oxide. Scattered angular fragments of quartz were

[6] The microstructure of some impure shales after burning has been described by Teetor, *Transactions of the American Ceramic Society* (1916), **18**, 843, but these were fired at Cone 3 (1190° C.).

[7] See Galpin: *Transactions of the American Ceramic Society*, **14**, 306–307.

also noticed. Biotite is rare. The hydromica and kaolinite in the shale fragments were arranged parallel to the bedding planes of the material.

*White Clay.*—This is very fine-grained, smooth, has a greasy feel, and high plasticity. The constituents seen under the microscope were light brown aggregates of very minute kaolinite scales, angular fragments of colorless quartz, together with small amounts of red iron oxide and occasional grains of brown biotite, yellow epidote, zircon, zoisite, and tourmaline.

### *Properties of the Clays*

The chemical composition of the two clays is shown by the following analyses:[8]

| | Brown | White |
|---|---|---|
| $SiO_2$ | 51.08 | 52.10 |
| $Al_2O_3$ | 22 05 | 31 54 |
| $Fe_2O_3$ | 10 43 | 2 38 |
| CaO | ..... | 0 10 |
| MgO | 2.28 | 0 58 |
| $Na_2O$ | 0.20 | 0.27 |
| $K_2O$ | 5 92 | 1.89 |
| $H_2O$ | 7.36 | 10.89 |
| | 99 32 | 99 75 |
| Total fluxes | 18.63 | 5.22 |

| | | |
|---|---|---|
| Amount of water required to develop plasticity | 44.0 | 47.7 |
| Plasticity | Fair | High |
| Air shrinkage | 4.5 | 8.1 |
| Color after firing | Red | Creamy white |

### *Effects of Firing*

*Color.*—The brown clay shows a gradual oxidation of the iron oxide, the color becoming darker and darker, though not strongly so, as the time of burning progressed.

The white clay shows no such pronounced change of color, owing to the small amount of iron oxide that it contains.

*Hardness.*—The longer the heating, the more compact and harder the clays became, due probably to progressive fluxing of kaolinite and hydromica with the cementing material, and with this there appeared to be a decrease in size of the pore spaces.

*Loss on Ignition.*—The driving off of chemically combined water or other volatile substances seems to have been practically completed, as was to be expected, during the stage of rising temperature (preheating

[8] We are indebted to Dr. T. L. Watson, State Geologist of Virginia, for permission to use these analyses in advance of their publication in a bulletin of the Virginia Geological Survey.

period) and so the amount of loss on ignition of the different samples drawn from the fire during the periods of constant temperature remains practically the same (Figs. 1 to 4).

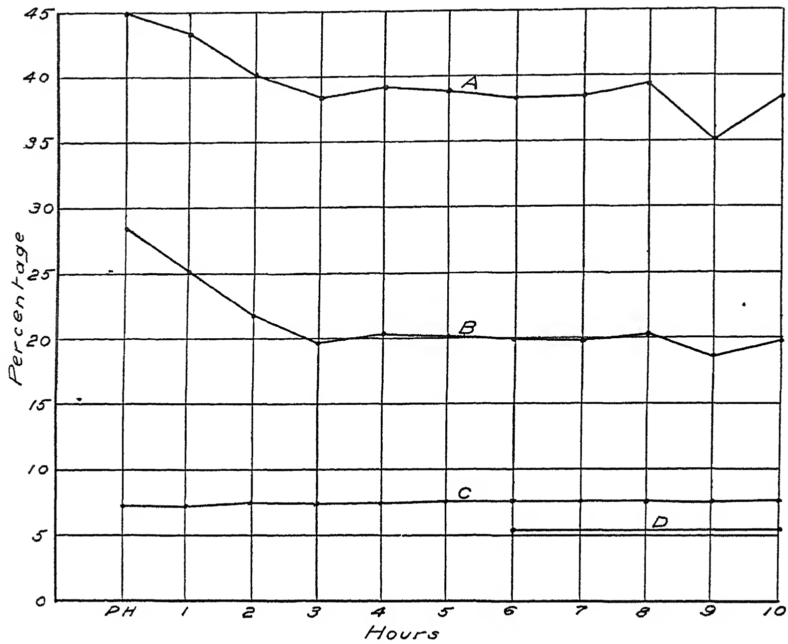


FIG 1.—BROWN CLAY FIRED AT 1000° C. A, POROSITY; B, ABSORPTION; C, LOSS OF WEIGHT ON IGNITION; D, FIRE SHRINKAGE.

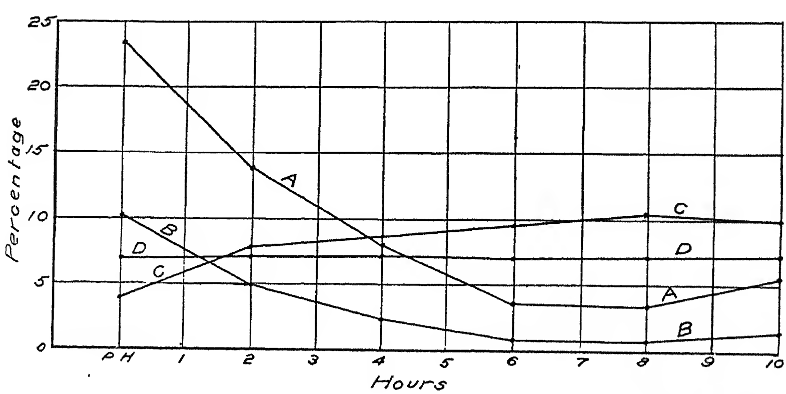


FIG. 2.—BROWN CLAY FIRED AT 1150° C. LETTERS ON CURVES SAME AS FIG. 1.

*Absorption and Porosity.*—The absorption was determined by soaking for 48 hr., while the porosity was found after a further boiling for 3 hr., this giving as good results as immersion in a vacuum. It will be seen by reference to Figs. 1 to 4, that the absorption and porosity curves run practically parallel with each other, the decreasing amount being a

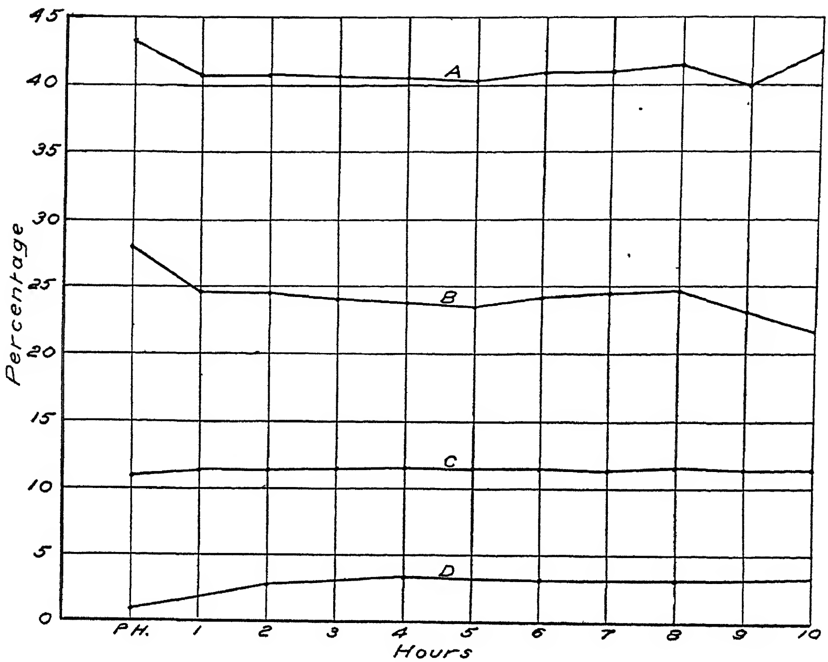


FIG. 3.—WHITE CLAY FIRED AT 1000° C. LETTERS ON CURVES SAME AS FIG. 1.

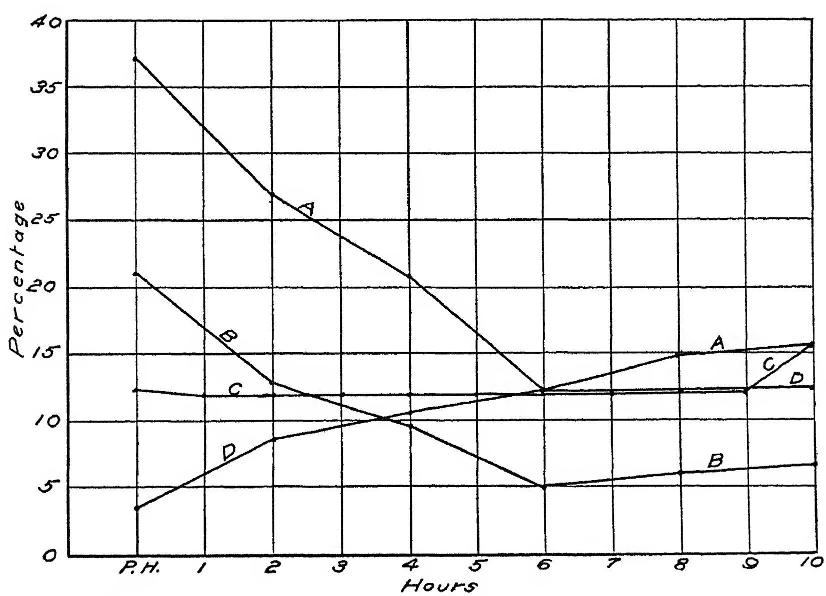


FIG. 4.—WHITE CLAY FIRED AT 1150° C. LETTERS ON CURVES SAME AS FIG. 1.

factor of temperature, due to the shrinkage of the pore space in the clays, and to fusion.

In the brown clay, both absorption and porosity decreased rapidly until the end of 3 hr., and after that they fell off very slowly.

The white clay behaved somewhat differently from the brown, there being no great change at 1000° C. but a rapid decrease in absorption and porosity at 1150° C. which decreased with the time of burning until near the end when it seemed to rise slightly.

As a matter of fact, both the absorption and the porosity decreased to a certain limit and then began to increase slightly in the tenth hour.

It is a little difficult to explain this increase in the final hour, for this clay does not appear to have reached its vitrification point (Figs. 1 to 4).

Tables showing properties of the burned clays.

TABLE 1.—*Brown Clay Fired at* 1000° *C.*

| | Color | Hardness | Loss on Ignition, Per Cent. | Absorption, Per Cent | Porosity, Per Cent | Fire Shrinkage, Per Cent |
|---|---|---|---|---|---|---|
| At the end of preheating | yellowish brick red | 2.5 | 7.17 | 27 89 | 44.95 | |
| At the end of 1 hr . | ↑ | 3.0 | 7.14 | 25.18 | 43 25 | |
| At the end of 2 hr . . | | ↑ | 7.47 | 21.77 | 40.09 | |
| At the end of 3 hr . . . | | | 7.40 | 19.60 | 37 92 | |
| At the end of 4 hr . . | | | 7.43 | 20 53 | 39.16 | |
| At the end of 5 hr. . . | darker | harder | 7 58 | 20.34 | 38.86 | |
| At the end of 6 hr . | | | 7.64 | 19.87 | 38.31 | 5.34 |
| At the end of 7 hr | | | 7 60 | 19.78 | 38.39 | |
| At the end of 8 hr. | | | 7.60 | 20.57 | 39 37 | |
| At the end of 9 hr . | ↓ | ↓ | 7 49 | 18.63 | 35.09 | |
| At the end of 10 hr. . . | brick red | 5.0 | 7.63 | 19.80 | 38.25 | 5.34 |

TABLE 2.—*Brown Clay Fired at* 1150° *C.*

| | Color | Hardness | Loss on Ignition, Per Cent. | Absorption, Per Cent | Porosity, Per Cent. | Fire Shrinkage, Per Cent. |
|---|---|---|---|---|---|---|
| At the end of preheating | light red | steel hard | 6.78 | 10.18 | 23 40 | 3.9 |
| At the end of 1 hr.. . . | ↑ | ↑ | 7 06 | | | |
| At the end of 2 hr.. . . . | | | 7.01 | 4.93 | 13 81 | 7.8 |
| At the end of 3 hr. . | | | 7 11 | | | |
| At the end of 4 hr.. | darker | harder | 7.15 | 2.26 | 7.90 | 8.7 |
| At the end of 5 hr.. . | | | 7.16 | | | |
| At the end of 6 hr.. . | | | 7.13 | 0.78 | 3.65 | 9.6 |
| At the end of 7 hr . | | | 7.17 | | | |
| At the end of 8 hr . . | | | 7.24 | 0.75 | 3.53 | 10.0 |
| At the end of 9 hr.. . . . | ↓ | ↓ | 7.22 | | | |
| At the end of 10 hr.. | dark brown | harder than steel | 7 28 | 1.49 | 5.53 | 9 9 |

TABLE 3.—*White Clay Fired at* 1000° *C.*

| | Color | Hardness | Loss on Ignition, Per Cent. | Absorption, Per Cent | Porosity, Per Cent | Fire Shrinkage, Per Cent. |
|---|---|---|---|---|---|---|
| At the end of preheating | slight pink | 3.5 | 10 86 | 27.95 | 43.19 | 1.00 |
| At the end of 1 hr... . | " | 4 0 | 11.37 | 24.62 | 40 67 | |
| At the end of 2 hr...... | " | ↑ | 11.40 | 24.53 | 40 81 | 2.83 |
| At the end of 3 hr... .. | " | | | 11 53 | 24.14 | 40.72 | |
| At the end of 4 hr... . | " | | | 11.58 | 23.82 | 40 46 | 3.47 |
| At the end of 5 hr...... | " | harder | 11.50 | 23.47 | 40.32 | |
| At the end of 6 hr...... | " | | | 11.53 | 24.16 | 40.87 | 3.17 |
| At the end of 7 hr... .. | " | | | 11.37 | 24.58 | 40.95 | |
| At the end of 8 hr.... . | " | | | 11.58 | 24.65 | 41.49 | 3.17 |
| At the end of 9 hr.. . | " | ↓ | 11.42 | 23.21 | 39.92 | |
| At the end of 10 hr . .. | " | 5.0 | 11.36 | 21.70 | 42.49 | 3.37 |

TABLE 4.—*White Clay Fired at* 1150° *C.*

| | Color | Hardness | Loss on Ignition, Per Cent. | Absorption, Per Cent | Porosity, Per Cent | Fire Shrinkage, Per Cent. |
|---|---|---|---|---|---|---|
| At the end of preheating | slight pink | 4.5 | 12.33 | 21 10 | 37.18 | 3.6 |
| At the end of 1 hr...... | " | harder than steel | 11.90 | | | |
| At the end of 2 hr.. . | " | ↑ | 11.85 | 12 76 | 26.75 | 8.5 |
| At the end of 3 hr...... | " | | | 11.93 | | | |
| At the end of 4 hr... .. | " | | | 11.93 | 9 63 | 21.00 | 10.6 |
| At the end of 5 hr...... | pinkish creamy white | harder | 12.00 | | | |
| At the end of 6 hr... .. | " | | | 12.01 | 4.95 | 12 31 | 12.3 |
| At the end of 7 hr...... | " | | | 12.01 | | | |
| At the end of 8 hr...... | " | | | 12.09 | 6.06 | 14.94 | 12.3 |
| At the end of 9 hr...... | " | ↓ | 12.21 | | | |
| At the end of 10 hr..... | " | harder than steel | 15.54 | 6.49 | 15 56 | 12.5 |

*Fire Shrinkage.*—The figures given in the tables show that in the series heated to 1000° C., the shrinkage remains practically constant after 3 hr. at that temperature, but that in the 1150° C. series the maximum shrinkage takes a little longer to develop.

Moreover, the fire shrinkage may show little change, even though there is a visible change in porosity, absorption, and hardness (Figs. 1 to 4).

*Tensile Strength.*—The tensile strength of the burned clays shows some interesting changes depending upon the time of burning.

The brown clay (Fig. 5) showed a tensile strength of 370 lb. per square inch after 5 hr. at 1000° C. and 430 lb. at the end of 10 hr. In the case

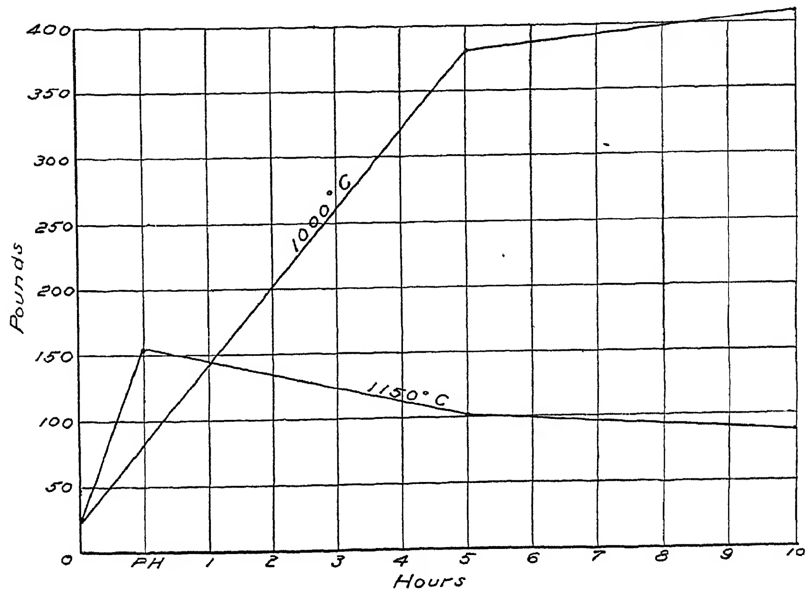


FIG. 5.—TENSILE STRENGTH OF BROWN CLAY AIR DRIED, AND AFTER FIRING.

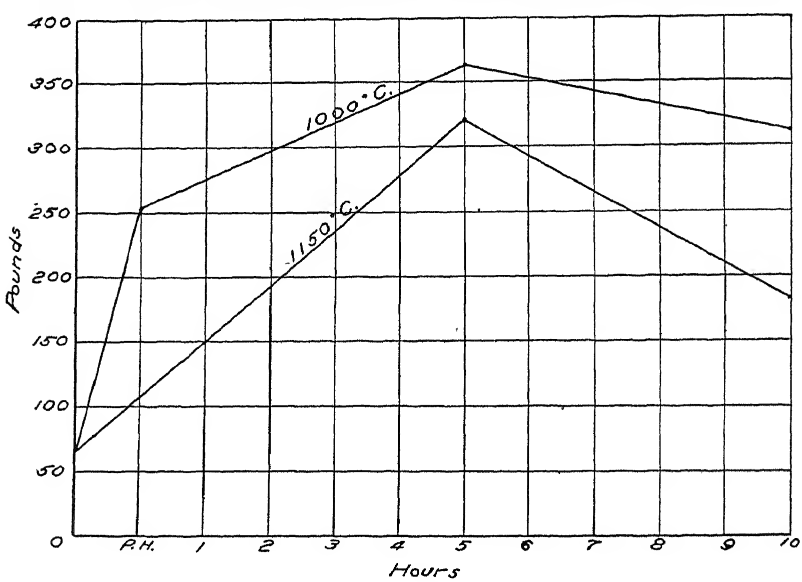


FIG. 6—TENSILE STRENGTH OF WHITE CLAY AIR DRIED, AND AFTER FIRING.

of the briquettes heated to 1150°, the tensile strength was 153 lb. pe. square inch when this temperature was first reached and then fell off to 80 lb. at the end of 10 hours.

The white clay shows a maximum tensile strength at the end of 5 hrs. heating at the same temperatures. The specimen heated to 1000° C. gave 360 lb. per square inch, while the one heated to 1150° C. showed 330 lb. per square inch.

*Microscopic Examination of the Burned Clays*

In order to study the pyrometric changes, a number of thin sections of the burned cubes were examined under the microscope.

*Brown Clay.*—Under the microscope this reveals what at first appeared to be a porphyritic texture (Figs. 7 and 8), but it was found that the phenocryst-like grains were small fragments of shale that had escaped pulverizing, while the groundmass is a brownish mixture of iron oxide with fine particles of kaolinite and hydromica.

These shale fragments are no doubt portions of the original shale, not completely decomposed, and hence they contain less free iron oxide than the other parts of the mass. They are much lighter in color than the groundmass and consist of minute grains of quartz, and small flakes of kaolinite and hydromica, the last two showing a distinct parallel arrangement, which is quite clear when viewed under the microscope, but does not come out very distinctly in the photograph (Fig. 7). The hydromica shows a higher interference color than the normal one and may be a sericite.

Small amounts of quartz are scattered through the groundmass.

At 1000° C. the specimens indicate no appreciable change in texture, but there seems to be a slight decrease of the scaly minerals and an increase in the amount of amorphous (isotropic) material toward the end of the burning.

The color as a whole became darker and the material seemed to be more tightly cemented in the specimens removed at the end of the firing period.

Vitrification seems to have begun at the end of 3 hr. exposure to 1150° C. and at this point the color changed to brownish-red and still later to reddish-brown.

The minute flakes of kaolinite and hydromica also seem to be disappearing, due in part to the fluxing of these minerals with the finer-grained ingredients, in which change the hydromica seems to flux more rapidly than the kaolinite. Their disappearance may also be due in part to their becoming isotropic under rising temperature. If so, the change was not complete at 1000° C.

In the shale fragments referred to above, there was noticeably less anisotropic material after 1 hr. at 1150° than there was after 1 hr. at 1000° C. Corrosion of the quartz fragments was also noticed at 1150° C., but it was less noticeable than in the case of the white clay.

When the thin sections are examined with polarized light, it can be

seen in passing from the specimens heated for 1 hr. at 1000° to those heated for 10 hr. at 1150° that there is a gradual increase in the quantity

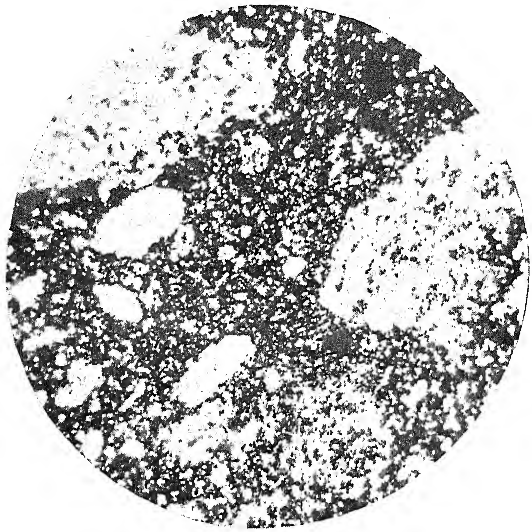

Fig. 7.—Photomicrograph of thin section of brown clay, after heating 1 hr. at 1000° C. Large grains, pieces of partially weathered unground shale. Black material mostly isotropic. × 50, with crossed nicols.

Fig. 8.—Same clay as Fig. 7, but heated for 10 hr. at 1150° C. Note greater amount of isotropic material as compared with preceding. × 50, crossed nicols.

of isotropic material. This is somewhat difficult to show in a photograph, as the halation produced by the grains still yielding light colors tends to

give the whole field a lighter tone. However, comparison of Figs. 7 and 8 will serve to show the contrast.


FIG. 9.—PHOTOMICROGRAPH OF THIN SECTION OF WHITE CLAY, AFTER HEATING 10 HR. AT 1000° C. Q, QUARTZ, SOMEWHAT CORRODED BY FUSION. DARK GROUND MOSTLY ISOTROPIC. × 50, WITH CROSSED NICOLS.

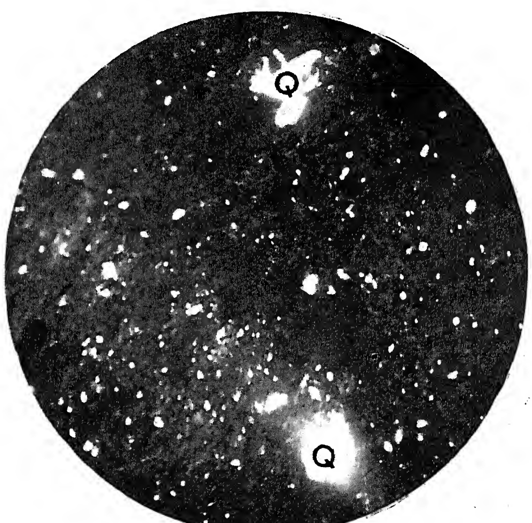


FIG. 10.—SAME CLAY AS FIG. 9, HEATED 10 HR. AT 1150° C. Q, CORRODED QUARTZ. × 50, CROSSED NICOLS.

As the fusion of hydromica and kaolinite proceeds, it results apparently in the formation of an iron silicate; volume changes also seem to occur, resulting in the development of pore space in the clays.

*White Clay.*—Thin sections of the fired material showed a somewhat banded texture which is not an original structure of the clay, but due to the pressure used in molding; they also showed small fragments of quartz scattered through the very fine-grained aggregates of kaolinite and hydromica.

Small grains of iron oxide were also noticed as well as tiny fragments of biotite, epidote, zoisite, zircon and tourmaline.

At 1000° C. the minute flakes of kaolinite and hydromica seem to be altered to a light brown amorphous body and the disappearance of these minerals seems more noticeable as the exposure to the temperature above mentioned continues.

Toward the end of the heating at 1000° C. and 1150° C., corrosive action of the quartz was quite remarkable (Figs. 9 and 10) and the change in color to a darker hue was also evident. At 1150° C., the gradual disappearance of the kaolinite and hydromica after heating 10 hr. at this temperature was very manifest. The edges of the quartz were more deeply eaten away according to the duration of burning, as shown in the figures.

TABLE 5.—*Mineralogic Analysis by Microscope*

BROWN CLAY

| | 1000° C. | | | | 1150° C | | | |
|---|---|---|---|---|---|---|---|---|
| | Iron Oxide and Isotropic Material, Per Cent | Hydromica, Per Cent | Kaolinite, Per Cent | Quartz, Per Cent. | Iron Oxide and Isotropic Material, Per Cent | Hydromica, Per Cent. | Kaolinite, Per Cent | Quartz, Per Cent |
| Raw clay..... | 40 | 28 | 27 | 5 | | | | |
| 1 hr. burning | 50 | 20 | 15 | 5 | 65 | 3 0 | 17 0 | 5 |
| 5 hr. burning | 55 | 15 | 15 | 5 | 76 | 1 0 | 5 0 | 5 |
| 10 hr. burning | 60 | 12 | 10 | 5 | 79 | 0 5 | 5 5 | 3 |

WHITE CLAY

| | 1000° C. | | | | | 1150° C. | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Kaolinite, Per Cent | Quartz, Per Cent | Hydromica, Per Cent | Iron Oxide, Per Cent | Isotropic Material, Per Cent. | Kaolinite, Per Cent | Quartz, Per Cent. | Hydromica, Per Cent. | Iron Oxide, Per Cent. | Isotropic Material, Per Cent |
| Raw clay.. | 63 | 25 | 10 | 2 | | | | | | |
| 1 hr. burning. | 53 | 25 | 10 | 2 | 10 | 19 | 25 | 1.0 | 2 | 50.0 |
| 5 hr. burning. | 35 | 25 | 8 | 2 | 30 | 9 | 25 | 1.0 | 2 | 60.0 |
| 10 hr. burning. | 35 | 25 | 8 | 2 | 30 | 9 | 25 | 0.5 | 2 | 60.5 |

In one sample which had been heated up to 1310° C., the corrosion of the quartz was very prominent, and at this point the clay seemed to show slight vitrification. However, no optical change was noticed in quartz up to this temperature.

In order to obtain some idea of the mineral content of the raw clays and the change in amount of each after heating, an attempt was made to estimate the per cent. of this material present. The figures given in table 5 are a rough quantitative estimate made from a study of the thin sections.

## Summary

From the foregoing data, we see that when fusion begins in the mass the fine mineral grains are first affected and seem to change from a crystalline to an amorphous condition.

There is a gradual change in the color as the time of heating at one temperature continues.

The hardness gradually increases on longer burning up to steel hardness.

Absorption and porosity in each case run practically parallel, decreasing to a certain limit and then increasing again slightly, later, in the case of the clays studied. Absorption and porosity abruptly decrease when the mass approaches vitrification, as shown in the case of the white clay at 1150° C.

Loss on ignition is the same at both temperatures, as might be expected.

The fire shrinkage in both cases shows no change in the last 5 hr. of continuous heating, while in the first 5 hr. of heating at one temperature it shows a marked increase.

The tensile strength seems to reach a maximum point after 5 hr. of continuous heating and decreases beyond that point.

Microscopic study shows that the longer the burning the denser the texture up to a certain point and that this is also accompanied in the brown clay by a change in color. There is, furthermore, in both clays a gradual increase in the amount of isotropic material.

Fusion of the hydromica and kaolinite proceed parallel with each other, but the former is more readily fusible.

Corrosive action as seen in the quartz increases with the length of burning.

Toward the end of the heating at 1150° C., pore spaces begin to develop, especially in the case of the brown clay.

# The History and Legal Phases of the Smoke Problem*

BY LIGON JOHNSON, NEW YORK, N. Y.

(St Louis Meeting, October, 1917)

ONLY the acute phase of the smelter fume problem is new. The problem itself is older than the Christian era.

While both lead and copper were mined and crudely smelted some 3000 years ago, it was not until the Roman occupation of the Iberian Peninsula and the British Isles, which occurred but shortly before the beginning of the Christian era, that there was any evidence of smelting operations on a scale sufficiently large to permit a fume problem.

Around Huelva, Spain, are found more than 30 million tons of slag from lead smelting conducted there by the Romans. Pliny tells us that more than 20,000 slaves were employed in the Iberian mines. Extensive mining and smelting by the Romans in England and Wales were coincident with the Iberian proceedings.

These metallurgical operations were upon a sufficiently large scale to produce marked results both upon the surrounding country and the smelter workers, but, as they were those of a conqueror upon conquered soil, conducted by slaves, imperial Rome failed to recognize that such a thing as a smoke problem did or could arise, and whatever fume question there may have been at that time remained a question only to those who had no chance to answer any phase of it.

The smelting operations of the Romans extended over about 400 years, and little is recorded of lead and copper smelting from that time until the 16th century. From the revival beginning at this period up to the present century, the growth of smelting has been comparatively gradual. In Great Britain, smelting was conducted almost wholly in localities where metallurgical operations were of paramount importance, and the communities that grew up in smelter localities were due to and dependent upon the mines and smelters. This fact had much to do with the comparative freedom of British smelting from burdensome fume litigation and legislation. To a lesser degree these conditions applied to the German operations which, when coupled with the further fact that most of the early German operations were to a greater or less degree fiscal workings of Prussia, Saxony and Brunswick-Hanover, accounts for

* Originally presented at a meeting of the New York section, Jan. 26, 1917.

the early immunity of German smelters. Topographically and with relation to fertile farm lands, the locations of the British smelters were superior to the German, and ultimately a strong legislative barrier was erected around German operations, particularly in the case of new smelters and under new locations of old plants

Nowadays, German smelting is, or at least was, prior to war conditions, under supervision of special officials in the nature of mining police and courts. Before construction of a smelter could be begun, application for a permit had to be made, accompanied by general data as to location, character, capacity, height of stack, character of ore to be treated and the like. Where a permit was once granted, it was not revocable, but notwithstanding the permit, if damage was done, the smelter could be required to change its methods and to install such appliances as would, so far as possible, prevent injury. Where a permit was refused, no smelter could be built. The refusal of a permit by the Bergpoleze, or mining police, was not final, but an appeal could be had to the Berg Gerict, or Mines Court, and finally to the Ober Berg Gerict, or Mines Court of Appeals, which was the tribunal of last resort. Incidentally, the Bergpoleze had inquisitorial or supervisory powers not only as to damage to vegetation but also as to the safety of the works and the life and health of the employees.

It is supposed by many that the British Alkali Act was aimed at the smelting industry. This is far from correct. The original acts were directed chiefly at the manufacture of soda, acids and ammonia. In 1861, the House of Lords provided for an inquiry "into the injuries resulting from noxious vapors evolved from certain manufacturing processes, and into the state of the law relating thereto." While the investigation covered smelting as well as the other enterprises, the resulting act (the Alkali Act of 1863) covered only so-called alkali works and provided for the elimination of not less than 95 per cent. of the hydrochloric acid produced in alkali processes. By an amendment of 1874, it was enacted that not more than 1/5 grain of acid should be contained in each cubic foot of gas escaping from the works.

The original act was made more comprehensive by amendments and revisions of 1874, 1881 and 1892 but, other than operations under the wet copper process and certain zinc processes, it was not until 1906 that smelting generally was included; and, even then, no actual regulation was provided, as the act so far as smelting was concerned merely declared that the inspectors under the act "may inquire whether in any works in which sulphide ores are calcined or smelted, means can be adopted at a reasonable expense for preventing the discharge from the furnaces or chimneys of such works into the atmosphere of any noxious or offensive gas evolved in such works, or for rendering such gas when discharged harmless or inoffensive" (Act 1905, Sec. 8 (1)).

No limitation was placed on smelting operations beyond inquiring as to the installation of such remedial methods or appliances as might be installed at a reasonable expense. There was no prohibition as to tonnage, and the first report of the Inspectors touching on smelting was in the 44th annual report of the department for the year 1907, 45 years after the passage of the original act, and this embodies no smelter regulations.

Fume controversies and the attendant litigation and legislation are of comparatively recent origin in the United States. Probably the first thing in the way of legislation in this connection was the ordinance of Oakland, Cal., enacted in 1872, prohibiting the erection or maintenance of any smelter within the corporate limits of that city. Subsequent California legislation was that of Contra Costa County limiting fume strength and by the State providing for a smelter commission to investigate the effect of fumes liberated in smelter operations. Efforts were also made to prohibit by legislative enactment the elimination of stack gases containing more than 0.2 per cent. sulphurous content. This proposed legislation, however, was defeated. There has been little legislation, either by the States or municipalities of the United States. Not until heap roasting of ores rich in sulphur was practised on an extensive scale, was there any litigation of consequence in this country, and to the condition resulting from heap roasting may be charged no small amount of the prejudice by the farmers against smelters and belief by them of the complete destruction of vegetation, animals and soil vitality through smelter operations.

The basis of this prejudice and belief is easily apparent, even to this day, in a visit to Butte, Mont., Shasta County, California, Ducktown, Tenn., and places where heap roasting was extensively practised.

Under heap roasting, strong fumes, in dense volume, were liberated directly on the surface of the earth. These fumes were heavier than air and cold, being practically of atmospheric temperature. They did not float off in the atmosphere but hugged the ground, rolling along in front of the wind in constant volume until absorbed by the soil and vegetation. There was little diffusion and the radius of fume influence gradually widened with the destruction of each successive barrier of vegetation.

The topography of the country surrounding the roast heaps was, almost without exception, sharply declivitous, as the roasting operations were conducted in hilly and mountainous regions. The soils of the regions surrounding the smelter sites were coarse and grainy, requiring the binder of the roots of vegetation and humus to hold the soil particles together.

The destruction of plant life adjacent to heap-roasting operations resulted not only in the loss of vegetal life above the soil but of the roots below, and, with every rain, the top soil was eroded until large areas were

made wholly barren, with the top soil washed away and only the decomposing rocks and mica left. In consequence, there was no soil in which vegetation could find a foothold, even when the fumes were no longer present. It was from this erosion that the belief of the killing of the soil first arose.

From what they could already see, the people around the smelters believed the smelter smoke to be the veritable breath of the Upas tree which destroyed all within its reach, and that this deadly agency was daily reaching farther and farther from the smelter and in and upon the homes and farms surrounding the smelter sites. Litigation was then not only threatened but instituted. First, the farmers sued the Tennessee and Ducktown companies, and upon the failure of these suits the State of Georgia took up the cudgel. The farmers in Shasta County, California, brought suit against the plants at Kennett and Coram; the Benecia residents and adjacent farmers sued the Selby smelter; the United States, by reason of the ownership of a forest reserve adjoining the Keswick plant, filed its bill against the Mountain Copper Co. Not long after this, the Deer Lodge Valley farmers filed their suits against the Washoe Co. at Anaconda, and Salt Lake valley farmers began proceedings against the smelter operations in that valley. The epidemic of smelter litigation was under way.

With these suits, developed the early so-called smoke experts. Most of them had been in the heap-roasting communities where the destruction of vegetation and soil erosion had followed the heap roasting. No definite knowledge of fume action was available, and the experts for both plaintiffs and defendants, in the absence of knowledge, began to form pet theories only to be met by pet theories of others directly at variance.

In the good old days of theory, an expert could make a casual inspection, or examine a few cross-sections of leaves under a microscope, or make a few comparative tests for the sulphur contents of healthy and unhealthy plants, and deliver an epic on what the fumes were and were not doing. The appearance of the country around the old roast heaps and early plants was too big a handicap for the smelters to overcome, and, with the notable exception of the Washoe litigation, in practically every case an injunction followed.

About the time of the institution of these suits, the smelters began the abandonment of heap roasting and the development of present day operations. In this development, prior theories expounded by the then experts or testified in the smelter suits had considerable bearing. In one locality everyone was assured that the injury came wholly from the dust particles; in another, from the dust particles as the nucleus for minute drops of acid; in another, from sulphuric acid vapor, which could be seen as a white fume cloud, and practically everything was blamed more than the chief, if not sole offender, the $SO_2$. In many cases the courts followed these theories in their decrees.

The first development went to long settling flues, smoke houses and dust chambers, cooling and slowing down gases, and taking out so far as possible the solids and acid vapors. At the Tennessee plants, acid making was attempted.

In most of the litigation, under readjustments to conform with decrees entered or modifications of these decrees—which modifications, as a rule, were secured under very substantial cash payments to the farmers—the smelters secured a new lease on life. Several, however, such as the Balakalala plant in California and the Highland Boy at Salt Lake, were completely closed, and passed into history as smelters. Most of the new installations, methods and readjustments provided by the decrees were in conformity with the theories of the experts in that particular case.

It is interesting to note that not only were the findings of fact in the different cases widely at variance, but the conclusions of law in many instances directly in conflict, even in cases in different circuits of the United States courts. For instance, in California and Montana, the United States court held that the court could consider the balancing of conveniences, that is that the court, in determining whether or not an injunction would be granted, could weigh the damage that would be done under closing a smelter against the benefits the farmers would receive under an injunction. In the federal court in Utah a directly contrary holding was made. In the findings of neither law nor facts did the courts coincide.

Some little time was required to make plant changes to conform with the theories of the experts adopted by the court or the farmers, and still further time was required to demonstrate the effectiveness of the changes. Most of the theories failed to pan out, and gradually a new crop of complaints, and claims that the recent installation did not prevent the damage, began to grow. Some claimants were insistent. Plant managers, remembering the results of old litigation, sought to temporize, instead of ascertaining the real facts and meeting them squarely. Some tried to buy peace.

Let me say just here that nothing in the Selby report is truer than the statement that (p. 14) "The policy that 'buys off' trouble, as the most expedient commercial method for abating it, has been responsible for much of the smelter litigation of this country and the intense ill feeling that unfortunately exists toward smelters in many smelter communities." It did not take long under this practice for the price of peace to rise to prohibitive figures and a new epidemic of litigation was threatened.

Going back a little beyond this second period of threatened litigation and before the storm clouds of it were yet clearly over the horizon, I think that I may say that the smoke problem was then considered by most plant managers as the least of their troubles. Little or no thought

or study was given the question, and when it was mentioned it was waved aside. The worst most of them feared was having to distribute judiciously a little peace money.

A number of the smelters, such as the Washoe, Mammoth, Bully Hill, the old Mountain Copper, the proposed Engles smelter and others, were either in or adjacent to National Forest Reserves and numerous reports of damage, or threatened damage, began to come into the Chief of the Forest Service and the Land Office. These reports were in turn referred to the Department of Justice. I was at that time Special Assistant to the Attorney General and these reports were finally referred to me with instructions to prepare and file bills for injunction unless some satisfactory solution could be reached.

While it was probably unknown to the smelters generally at that time, it was never the intention of the Government to close any plant. What it sought was the immediate and serious consideration of the fume problem and active effort to correct harmful conditions where such existed. Only one suit was filed, and that was against the Anaconda plant. In each of the other cases stipulations were entered providing for research work and the installation of the highest types of methods and appliances known to smelting science or else the abatement of the operations producing fumes in harmful quantities until such appliances had been installed and proven elsewhere, upon which they were to be installed at the smelter entering into the stipulation. In the Anaconda case the first scientific commission was agreed upon. We proposed that the smelter operate under the best type of methods and appliances known to science and commercially feasible at the plant and that a commission of John Hays Hammond, Dr. L. D. Ricketts and Dr. Joseph A. Holmes, Director of the Bureau of Mines, be designated as a commission to investigate and prescribe what changes or installations should be made under the agreement.

It was not long after this that the threat of further litigation by the farmers became ominous, and, as I had completed my work with the Department of Justice, so far as smelting matters were concerned, I resigned to become what might be termed smoke or field counsel for the American Smelting & Refining Co.

The first urgent smoke matter in this connection was the Selby litigation between Benecia residents and the Selby plant, which had been pending a dozen or more years. A flat injunction against the operation of the Selby plant had been granted, which injunction had been fought through the courts and finally approved by the Supreme Court. The case had been through the court of last resort and the injunction was to become effective in a month or 6 weeks' time.

On examining the record, I found this suit to be somewhat different from the average smelter suit. Very little claim of injury to vegetation was made. More stress was placed on injury to animals, but the chief

ground of complaint was excessive discomfort and alleged nausea and illness produced by odors which the witnesses uniformly declared "to smell like rotten eggs" or the sulphur spring of an adjoining county which gave off quantities of sulphureted hydrogen.

An examination of the Selby plant disclosed that, regardless of what lead loss there might have been at other times, under changes made or under way no lead loss would occur and therefore no possible source of injury to animals could exist from the plant's operations and that there was no appreciable amount of sulphureted hydrogen given off by or generated in the smelter plant.

In the San Francisco region, in the spring and summer months constant trade winds blow, and these winds follow a direct line from the Selby smelter to Benecia where most of the complaining witnesses lived. Beyond the Selby plant, but also directly in line of the trade winds over the Selby plant, was a large oil refinery and asphalt works. The odors complained of by the witnesses all came in the early morning hours, and investigation disclosed that at the times involved in the testimony, the oil and asphalt stills were not capped and were poured at three or four o'clock in the morning. Here, then, was a source of odors which traveled on the same winds that carried the Selby smoke. The smelter was guilty of none of the things upon which the injunction was based. Capping the oil and asphalt stills practically disposed of the unpleasant smells.

We notified the County Attorney and Commissioners of Solano County of these facts and said that we proposed to decline to observe the injunction prohibiting the operation of the plant. This would have meant additional long drawn out and expensive litigation. To obviate such a situation, I proposed that the whole matter be referred to a commission of scientists of the highest type, the members to be agreed upon jointly, and the finding to be entered in the Selby case as the findings and decree in that case. Every access to the plant and fullest facility for examination was to be given the commission. After many conferences, an agreement along these lines was arrived at and the Selby Commission came into being. The members of the Commission were Dr. J. A. Holmes, Director of the Bureau of Mines, Dr. E. C. Franklin, then Director of the Chemical Laboratories of the U. S. Public Health Service at Washington, and Ralph A. Gould, a chemical engineer of San Francisco. With the Selby Commission came the first scientific research in smelter fumes along the lines of normal field conditions.

The examination of the Selby Commission extended over a period of about a year and a half and in the end clearly demonstrated that the smelter was doing none of the things found against it in the original decree; and the injunction was vacated.

Shortly after the time the Selby Commission began on its work, the murmuring of discontented farmers became much louder. Crops had

failed in many localities and wherever this crop shortage was in a smelter region the smelter was blamed. This was particularly true where prior payments had been made to farmers. Peace was quoted at war prices.

As I before stated, this was very shortly after the Selby Commission began its work. Other than the research just beginning under the Selby Commission, there had not been in the United States or elsewhere any experiments or research work with smelter fumes or $SO_2$ under natural field conditions. It is true that Haselhoff and Lindau, Von Schroeder and Reuss, E. Schroeter, Wieler and Hartlieb, R. Hartig, Wislicenus, Freitag, Sorauer, Ramann, Gerlach, Schmitz-Dumont, Sabachnikoff, Raubner, and Stockhardt in Europe, and Haywood and Pierce and, to to a limited extent, Ebaugh and Talmage, in the United States, had carried on some experiments, but in none of these were normal field conditions approximated. Some fumigations were carried on with leaves or twigs or small parts of plants in bell jars. Others were in hermetically sealed cabinets or "smoke houses" so constructed that only abnormal conditions could result. The plants used were grown in pots or flats and were grown and kept under conditions that were not normal. The fumes introduced were from burning sulphur or sulphur and alcohol, or else concentrated $SO_2$ let in from a tank or gas burette. The concentrations were unknown, absorption was not considered nor were chemical analyses of the air made. Environmental factors of temperature, humidity, light values, barometric pressure and time element of exposure were not considered. The plants used in the experiments were not grown to maturity, nor were their food or crop values ascertained. As a matter of fact, at the time these experiments were conducted, no quick and accurate method of air determination was even known to the experimenters. Even had these investigations been conducted with normal plants, under natural field conditions, the lack of knowledge or records of actual fume strengths reaching the plants and of the environmental factors involved would have rendered the experiments practically valueless.

Some investigators of smelter regions placed much reliance on the sulphur content of plants collected in smelter neighborhoods, notwithstanding the fact that investigations in the Department of Agriculture have shown that the sulphur content of the same plant may vary as much as 3000 per cent. at different times during the season and at the same period the content of the same species of plant will vary markedly, particularly under different percentages of soluble sulphur in the soils, which frequently occurs at short distances.

In other words, from farmers and their experts the smelters were coming in for full blame for all crop failures and the managements actually did not know whether or not they were doing the damage. From my past experience, I knew that there were numerous plant injuries attributed to smelter smoke which even the most competent expert could not differ-

entiate by mere observations or even microscopic study. Numberless diseases and injuries which were not caused by smelter smoke were pointed out as evidence of smoke damage.

No plant manager knew how much, if any, of the damage attributed to the smelter was caused by it, and if there was damage, what operation of the smelter or element of waste had produced the damage. And, too, with the payment of damages communities of smoke farmers grew up. Their idea of farming was to let their places grow up in weeds and collect from the smelter the value of the maximum crops ever produced. Many smelters had paid damages for conditions which, it later developed, were not remotely attributable to smelter operations; but the farmers once having been paid insisted on continued payments. The price of peace was rising above the possible profits of smelter operation.

The time for theorizing had passed. It was necessary that the smelter managements should know just what damage was being done, and if there was damage, how it was done and what was necessary to prevent it. Under my urgent recommendation, the first research department to ascertain all this was installed and, in conjunction with the Selby Commission, it blazed the way for the smoke engineering.

This research work involved chemistry, plant pathology, plant physiology, entomology, agronomy, dairy husbandry and veterinary toxicology and meteorology.

You may wonder why all this was necessary just to find out whether or not the smelter was doing any of the things claimed, and why one or two experts familiar with the appearance of vegetation in smelter regions would not have sufficed.

We had to know first what smelter fume or parts of fume would do damage; how the injury could be caused; what conditions were or could be confused with smoke injury, and finally, where damage was or could be done, what steps were necessary to obviate it.

This meant extensive research work under normal field conditions to ascertain the effect of smelter eliminations upon vegetation and animal life. The time at hand is too short to go into minute description of this research work. It will probably suffice to say that we found that dust and acid vapors were practically negligible quantities in the fume problem so far as vegetation was concerned. Where damage was done, this could be traced almost if not wholly to sulphur dioxide. Treating plants with the amount of solids that would be deposited on vegetation under general smelting operations produced no result. The average vegetation can stand fifty times the strength of acid vapor that it can resist where the sulphur is administered in the form of $SO_2$. In the ordinary smelter fume the dust and acid vapor are but a small fraction of the sulphur content, the latter being chiefly confined to sulphur dioxide.

It then became necessary to ascertain the relative resistance of the various plants to sulphur dioxide and the conditions under which injury occurred when the $SO_2$ was present in sufficient quantities to do damage.

To develop this fact it became necessary to know exactly how smelter fume acted upon plant life, how it obtained ingress to the plant structure and its effect after its absorption in the plant.

It may not be amiss just here to describe in a few words how this ingress comes about and the results which follow. In the beginning we found that only during the period that the plant was in leaf could injury occur. Subjecting a plant to $SO_2$ or other fume while not in leaf produced no result. In most instances, and practically without exception in alkaline soils, we found that treating the soil with sulphur or sulphuric acid—that is, dusting or spreading crude sulphur upon the soil or spraying the soil with acid—resulted in increased crop yield. Sulphur administered in this way was a benefit and not an injury. This conclusion was verified by independent experiments conducted at the University of California in Berkeley, at an experiment station in Oregon and at the experimental farm of the Anaconda Copper Co.

The question then narrowed down to the effect of smelter fume through leaves and on leaf structure and to present this clearly it is necessary to describe the ordinary leaf and its functions.

All leaves under normal field conditions, in climates such as we have to deal with here, have an epidermis covering both under and upper leaf surface. This epidermis is impervious to moisture and for all practical purposes, except where stomata, or breathing pores, occur, is also impervious to gases. These breathing pores are so small that considerable magnification under a microscope is necessary to see them at all. The stomatal openings are entirely too small to admit the minutest suspended solids, water globules or particles of mist or vapor. The stomata at the leaf surface are faced with two guard cells, that are also covered by practically impervious epidermis. These guard cells, under certain leaf functions open or close the stomatic chamber. With the guard cells closed, the leaf under ordinary conditions presents impervious surfaces to the elements. The inner content of the leaf, or the mesophyle, is composed of palisade cells, collecting cells and sponge parenchyma, in which, under the action of light, starch and sugar are formed and plant food manufactured and supplied to the plant.

We next found that under conditions of absolute closure of the stomata the plant was a number of times more resistant to $SO_2$ than when the stomata were open. Under all ordinary conditions the stomata form the points of ingress of gas. It followed, therefore, that where an external condition brought about the closure of the stomata it brought about increased resistance, and we finally worked out four factors in chief in fume injury, where sulphur dioxide is present in strengths sufficient at any

time to bleach plant life, these factors being light, humidity, temperature, and constant wind direction. The bearing of the latter comes from the fact that injury, except from abnormally strong fumes, does not come from mere fume puffs but from steady application of $SO_2$ for several hours. The other three factors bear upon stomatal opening.

After we had learned the strength of $SO_2$ necessary to produce markings upon plant life under field conditions, it became necessary to know whether or not $SO_2$ was present in such quantities in the field. It was also necessary to know accurately the wind constancy and direction, the limits of the smoke stream, temperature, humidity, and general weather data. Complete installation of standard meteorological instruments, under the supervision of the local U. S. Inspectors was made; and portable laboratories, that is, laboratories set up in small automobiles which could follow the smoke stream as well as make general tests, were secured. This was in addition to the fixed air stations maintained. Under the method of air analyses devised by J. R. Marston and later refined by A. E. Wells, chief chemist for the Selby Commission, quick and accurate determination of air samples was possible, and, more important still, these samples could be taken in a few seconds time, and, in this way, the constancy or inconstancy of the $SO_2$ determined. For the smelter plants, we cross-sectioned the several flues at the base of the outlet stacks, and the stacks, to afford analysis of their contents, and installed self-recording thermometers at appropriate points in the flues and stacks for accurate and complete data as to gas temperatures.

With this knowledge, the next step was to ascertain with scientific exactness the cause of crop failure or injury where the appearance of external conditions, and often even microscopic cross-sections, were seemingly identical with the results of smoke injury, but where our other investigations had developed that smoke injury could not have occurred. There are many diseases, pathological conditions and insect injuries which, both externally and under the microscope, present an appearance practically identical with that of smoke injury. With the completion of this investigation, a survey of the entire so-called fume zone followed.

It is interesting to note that, in determining the injury of plants by disease, we can establish our diagnosis with scientific exactness not possible in the diagnosis of any disease of human beings or animals. We can establish our diagnosis beyond the possibility of doubt or question. This is done by what is known as the pure culture method. The unhealthy plant is pricked by a needle-pointed instrument which is then dipped into a special and pure plant jelly, and a culture of the disease organism is grown. The disease organism is isolated and reproduced. A healthy plant is then inoculated with this culture and, after the disease develops, comparison of appearances is made; a second culture is made from the inoculated plant and a second inoculation of a healthy plant follows. In

this way the disease organism is isolated, the disease and appearance reproduced and the diagnosis established beyond a reasonable doubt. Where culture media are impractical, resort to direct inoculation is not only possible but convincing in its results. For insect causes, we find and collect the causative insect. As some insect colonies, although numbering thousands or even millions, are so small that the individuals are observable only under very high-powered microscopes, and others work wholly out of sight and beneath the soil, it requires a competent entomologist to handle this phase.

It will suffice to say here that we proceeded through the whole list of plant pathology, physiology, entomology, agronomy and dairy husbandry and veterinary toxicology with equal care and certainty. The time is too short to go into detailed explanation of the plan pursued under each. Great credit is due to P. J. O'Gara, chief expert in charge, and E. P. Fleming, chief chemist, for the perfection of means for and methods of the scientific research done as well as for the results accomplished.

One other agency in smoke determination has been established with relation to plant life, and that is the evidence of so-called guide plants. There are certain plants, such as barley, which are especially susceptible to fume injury and will show bleaching long before the possibility of injury to other plants. Where the guide plants are unmarked, it is self-evident that the trouble with adjoining hardier plants is not smelter smoke.

We also carried crops to maturity after producing severe burns on some and repeated burns on others, with astonishing results as to crop yield. We found the real damage from the average burn more imaginary than actual. We ascertained the protein content of bleached plants, finding it as high or higher than in the unbleached. We proved food values by actual feeding. In fact, we covered every phase we could find.

In the work I have mentioned, we not only secured information valuable to the smelters but we accomplished actual and beneficial conservation so far as the agricultural resources of the country were concerned.

Scientific farming is not yet widely practised. Farming methods are often handed down from father to son. Year after year the same crops are planted, often on the same soil. This applies to farming generally and not merely to smelter communities.

In many places not only do the soils become impregnated with disease-producing fungus and germs, but the result becomes so widespread that no healthy seed can be secured in the community. It often happens that disease conditions may largely be eradicated by proper treatment and planting of the seed and crop cultivation, but these facts, as well as that the seed is diseased, are unknown to the farmer.

I recall a case where enormous claims for loss of potato crops were

filed against a smelter. Upon investigation we found both the potatoes and the soil of the region highly infected with two of the most disastrous of potato troubles, rhizoctionia and fusarium. A subsequent search of the seed stores in the community failed to disclose a single healthy sample of seed. Naturally the potato crops were failures. The sad part of it was that there was no need of this loss. Healthy seed procured elsewhere and planted in soil where potatoes had not been grown for several years would have given the old-time record crops, and proper treatment of the local seed would have produced infinitely larger returns.

Most of the diseases and insect conditions of agricultural communities are largely preventable, and for those that are not steps may be taken to greatly reduce the harmful effect. Much of the trouble can be directly pointed out to the farmer, and the means of eradicating or lessening these troubles shown to him. This, of course, involves the demonstration that the unhealthy crop conditions are not attributable to the smelter.

The farmer who follows the suggestions for eradicating unhealthy conditions soon finds his crops exceeding those of his neighbors who have failed to observe the same precautions. In this way, gradually, it is true, the influence of the smelter's experts is felt, a friendlier feeling is engendered, agricultural conditions are improved and actual conservation accomplished.

This brings us down to one final theory which in some places is still firmly entrenched and threatens possible trouble. I refer to visual clearance and the belief that anything seen coming from a smelter stack should be labeled with a skull and cross bones. Our investigations showed that the visible fume was little to be feared as an actual instrument of damage to vegetation. Under former operations, where a good part of the lead content of the ore was blown up the stack and where heavy arsenic fumes were given off, this metallic loss had some connection with certain animal troubles, but had little effect on vegetation. Where this character of loss was not involved, the visible fume was and is chiefly dangerous from the psychological point of view. By this I mean that most farmers (and many will be honest in their belief) will assert, so long as there are visible stack eliminations, that every crop failure is due to the smelter fumes. Therefore, to meet this, it will be advantageous to obtain the greatest possible clearance up to the point—and I want to stress this especially—where the efforts to secure visible clearance renders the smelter operations more liable to do actual damage. Beyond this point it is better to try to educate the farmer and get rid of the belief. Just because a farmer sees dense volumes of soft coal smoke pouring from the stacks of manufacturing plants, business houses, engines and the like, he feels no uneasiness about his crop. There is little, if any, more danger to crops in the visible smelter smoke. And, too, even where the belief of damage is deep seated, mere belief can be met by facts, but

actual damage cannot be dodged because the smelter management seeks to meet some psychological condition; for mere belief will not last long in the face of actual damage.

Get clearance first for actual recoveries, and to prevent any excessive loss of toxic solids, and next, so far as possible, to meet any psychological conditions, but do not place visual clearance above the likelihood of actual damage. As I have previously stated, the visible part of smelter fume is not the harmful agent.

Our investigation and the temperature curves we have worked out have shown us that where there is fume injury the answer to the problem is in hot gases, gas dilution, and high stacks. Anything that cools or slows down the gases, or results in their liberation at low altitudes, invites, to say the least, the probability of actual damage. Promising a community complete visual clearance, or encouraging the belief that visual clearance means impossibility of damage, is borrowing trouble for the future, for such a clearance, unless I am mistaken, means low gas temperatures.

These smoke problems are bringing to the front a new specialist, the smoke engineer. The far-sighted smelter management, under existing conditions, for all new constructions, and even alterations and additions to old plants, will find this new expert a necessity rather than a luxury. The best way to take care of trouble is to prevent its starting. Many of the smelters realize this and welcome the coming of the smoke engineer and specialist.

## Discussion

P. J. O'Gara,* Salt Lake City, Utah.—I have had charge of the investigations of the smoke problem for the American Smelting and Refining Co. during the past four years. One of the worst burns we ever experienced occurred shortly after we began putting all our gases through the baghouse. This occurred in 1914 after the roaster gases had been put through the baghouse; previous to that time, the greater portion of the roaster gases was not put through the baghouse.

Everyone knows why a hot-air balloon goes up. The hotter the air in the balloon, the higher it will go. In other words, hot gases will rise higher than cold gases. When gases are handled in any way that will tend to lower their temperatures, these gases are going to travel less high in the atmosphere and will therefore tend to reach the ground sooner than if they are turned out very hot.

So far as injury to vegetable life is concerned, sulphur dioxide is the only element in smelter smoke that need be considered. The solution of the problem will be either to take out the excess of sulphur dioxide

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or to diffuse it high enough up in the atmosphere so that the concentrations, when it reaches the ground, will be so low as not to cause injury. Taking the visible elements out of the smoke stream will not solve the problem. In other words, filtration by means of a baghouse will tend rather to increase the possibility of injury to vegetation than to lessen it, for the reason that the baghouse must be operated at relatively low temperatures as compared with the Cottrell or electrostatic process of precipitating the visible elements in the smoke.

In lead plants at which baghouses are now in use, and the possibility of injury to vegetation by sulphur dioxide must be considered, the best plan would be to Cottrell the roaster gases and permit the blast-furnace gases to go through the baghouse; blast-furnace gases contain but little sulphur dioxide and are cold as compared with roaster gases. It might even be a good plan to consider the advantage of keeping the gases or smoke streams from these two sources unmixed, sending them out from separate stacks, as is now the plan at the Murray, Utah, smelter. When removing solids from gases, consideration need only be paid to taking out the values. Absolute clearance should not be considered, inasmuch as this does not solve the sulphur dioxide problem but rather increases the possibility of injury.

So far, we have considered only the matter of diffusing the injurious element of the smoke into the higher atmospheres. There is still another solution of the problem, which it may be necessary to apply even in plants where the diffusion of sulphur dioxide into the higher atmospheres is being accomplished by conserving the potential energy in the gases (heat). It is well known that the amount of sulphur that may be eliminated from any smelter without injuring vegetation cannot go beyond a certain figure, no matter how much care has been taken to provide high stacks and high gas temperatures. When this limit has been reached, the only remaining remedy is to remove the excess sulphur from the gases, either as sulphuric acid, sulphur dioxide, or elemental sulphur. At Tacoma, liquefied sulphur dioxide is being made, taking it out of the roaster gases. At Garfield, a sulphuric acid plant is in operation.

Before considering what should be done to solve the smoke problem at any particular plant, the topographic features of the district should be known as well as the climatic conditions and the character of crops raised. The concentration of sulphur dioxide necessary to produce injury to vegetable life is very low, considering the concentrations that are found in the stack. We have determined that one part of sulphur dioxide per million, by volume, if applied to the least resistant crops, such as alfalfa, barley, and oats, for 3 hr., with atmospheric humidity over 70 per cent., and temperature above 40° F. during daylight, will produce the first visible signs of injury. If the concentration of sulphur

dioxide be increased only 10 per cent. the injury will be quite marked, showing that when the limiting concentration of sulphur dioxide has been reached it takes but very little more to produce well-defined injury. In any district, only the least resistant drops need to be considered.

We are entering into a new phase of agriculture which will surprise some of you who are interested only in mining and smelting. In our experiments we, as well as others, have shown that sulphur applied to the soil as elemental sulphur or sulphuric acid is not only not injurious but actually beneficial, greatly increased crop yields being produced by this application. Even sulphur dioxide, in such low concentrations as not to produce visible injury to vegetation, will have a beneficial effect, as noted in the slightly increased yields and the higher protein content of the crops subjected to this gas. Highly alkaline soils arê particularly benefited by the addition of sulphur and sulphuric acid, the tendency being to reduce the water solubility of the alkali. We have also found that the potash and phosphorus in soils treated by sulphur and sulphuric acid are rendered more available to growing plants. Beneficial bacterial reactions are also stimulated. Where sulphur is applied to soils that tend to be naturally acid, lime also should be added. The problem of supplying sulphur to soils, either alkaline or acid, is not difficult. In our experiments, sulphur at the rate of 200 to 400 lb. per acre has caused enormously increased yields of a large number of crops; likewise, sulphuric acid, with the above sulphur equivalent. Much valuable work has already been done by certain agricultural colleges and experiment stations in the States of Wisconsin, Iowa, Oregon, California, Montana and a number of others. It has been pointed out that sulphur is of particular importance in the production of leguminous crops such as alfalfa, clover, peas, beans and the like. Other crops, such as the cereals and sugar beets, have shown increased yields where the soil was fertilized with sulphur. It is apparent, therefore, that we are going to have an outlet for sulphur through the channels of agriculture. Crops must have sulphur and many of our soils are actually sulphur poor. The protein in our cultivated plants contains, as an average, about 1 per cent. sulphur, which must be supplied by the soil. The amount of sulphur demanded by the various crops may range from 1 lb. of sulphur per dry ton of vegetable matter to as much as 20 lb. of sulphur per dry ton. It is evident, therefore, that in time soils become deficient in sulphur as they do in potash, phosphorus or other elements, the compounds of which are utilized by growing plants. Great areas of our intermountain and Pacific coast soils contain enormous amounts of locked up potash and phosphorus and we have found that by adding sulphur or sulphuric acid the potash and phosphorus availability is increased. The availability of potash and phosphorus has been increased fully 20 per cent. above that of similar untreated soils

by the addition of 200 to 400 lb. of sulphur, as such, or its equivalent in sulphuric acid. The bacterial activity in the soils has been stimulated fully 50 per cent. in 3 or 4 weeks' time by the same treatment.

I am not a smelter man at all, but rather a botanist. I perhaps know the difference between a blast furnace and a roaster, and I also know that certain of the products from these furnaces are valuable for the agriculturist, and I fully expect that the smelters are going to be the best friends of the farmer. Do not try to fool the farmer (or yourselves) by simply clarifying your smelter smoke, but explain to the farmer that sulphur is actually beneficial and that he needs it. Sell the sulphur to him at the right price and he will take it and increase his crops.

# Zinc Dust as a Precipitant in the Cyanide Process

W. J. SHARWOOD, LEAD, SOUTH DAKOTA

(St. Louis Meeting, October, 1917)

In the cyanide process, gold and silver are dissolved from crushed ore as double alkali-metal cyanides, from which they may be precipitated by such positive metals as sodium (amalgam), aluminum, or zinc, or by electrolysis. Two extreme conditions may be noted. Some works, especially slime plants practising decantation, use a relatively large volume of solution—possibly 4 or 5 tons per ton of ore—nearly all of which may require precipitation, so that the solutions handled are of much lower value per ton than the ore. On the other hand, in some leaching plants it is possible to extract with very little solution, and to percolate some of this more than once through the charge before precipitation, so that the solution to be precipitated may be much less than half the weight of ore, and proportionally richer. Solution intended for further use need not have all its precious metal removed, but any that has to be thrown away should be impoverished as far as is economically possible.

In spite of certain advantages possessed by other precipitants, zinc in some form has been almost universally used. In some of the first attempts to utilize cyanides as gold solvents, a "piece or plate of zinc" was suggested as a precipitant, but extension of surface was early recognized as a desideratum.

Macarthur and the Forrests adopted a "metallurgical filter" of zinc shaving, turned from disks or rolled sheets. They had previously experimented on other forms of zinc, and Macarthur records having tried zinc dust, which had of course been long known as a general reducing agent. It had also been known and used as a precipitant of precious metal from plating and photographic solutions, and comminuted zinc had been patented for recovering copper, etc., from ore leaches. Other inventors proposed the virtual making of zinc powder by the attrition of balls of zinc, and by similar means, during the passage of a stream of gold-bearing solution.

Sulman claimed the use of zinc dust, or fume, in a special apparatus, effecting a more or less regular feed of dust and solution by means of intermittent siphons; the mixture or "emulsion" (as it is still called) rising with diminishing velocity through an inverted cone in which it

deposited most of its burden, and being clarified by passing a baffle-box and finally a cloth filter. This system was used at Deloro, Ont.

The first cyanide plants in the United States to use zinc dust were the Mercur (Utah), treating coarsely crushed oxidized or roasted ore, the Drumlummon (Marysville, Mont.) and Delamar (Idaho) tailing plants, followed by the Homestake Sand Plant No. 1 (South Dakota). All these treated large tonnages and used filter-presses to catch the precipitate; the first three, and the Homestake when first installed, used square presses of the Johnson type, but differing in size and design. Nearly all the larger cyanide installations on the American continent now precipitate with zinc dust, and the Merrill triangular press has almost entirely superseded other forms for this purpose. Zinc shavings are still used almost exclusively in South Africa, and in the smaller plants elsewhere. Aluminum dust is used to a limited extent, chiefly on Canadian silver ores.

## Zinc-dust Precipitation

The early method of applying zinc dust was to fill a vat with pregnant solution, agitate, add the dust and pump through the filter press. The zinc was first introduced by stirring the body of solution mechanically and sprinkling the dust on the surface. A variation was the use of compressed air at the vat bottom, through a small central cross or coil of perforated pipe. In a few seconds this set the solution in violent motion, and the zinc dust was then scattered on it with a shovel. This was at best a dusty and disagreeable job, and the introduction of oxygen at the precipitating stage was opposed to chemical theory. Elimination of the air, however, and cautious sifting of the zinc dust over the surface before pumping, showed imperfect precipitation—for instance, some tests gave only 75 per cent. of the gold precipitated as against 96 per cent. with thorough agitation by air.

To obviate the use of dry dust, various attempts were made to add it in the form of an "emulsion" or suspension in water or in cyanide solution.

An inverted 3-ft. iron cone was fitted with a short valved hose at the apex; the base was covered with a plate having a covered handhole, two valved inlets for compressed air, one for solution, and a relief valve. This pressure cone was two-thirds filled with solution, followed by zinc dust (say 40 lb. for a 200-ton tank); compressed air was then admitted by an inlet leading nearly to the apex to stir the mixture while adjusting the cover. The relief valve was then closed and the "emulsion" sprayed through the hose on the surface of the pregnant solution, which had meanwhile been stirred with air in the usual way.

At one plant such precipitate as settled on the tank bottom was allowed to accumulate there, while only the suspended portion went to the filter-press. Two grades of precipitate were thus produced, the settled material being decidedly lower in value and hard in texture.

Elsewhere, when the solution was pumped nearly to the tank bottom, a man entered the vat in rubber boots and swept settled material toward the pump intake, a small amount of fresh solution being used to assist the sweeping out. In spite of this sweeping there was a tendency for hard lime-zinc scale to accumulate on the bottom, and to some extent on the tank staves, so that after 6 months there might be several thousand dollars thus tied up in a pair of large vats, only recoverable by periodical "scaling" in which hammers and chisels were used.

Pumping a large tank to the press might occupy 2 to 6 hr., and one might expect re-solution to take place during this period when the zinc was added all at once at the start. Repeated tests showed that, although the precipitation by mere agitation was far from perfect, very little re-solution took place during pumping, but the part played by the zinc accumulated in the press was evidently an important one.

The amounts directly precipitated ranged from 20 to 90 per cent., while after passing the press 92 to 98 per cent. of both precious metals had been removed, but only 1 to 4 per cent. of the copper. Recently the positive removal of oxygen from the entire body of pregnant solution before adding zinc has been carried out on a working scale. Long-continued tests show that, by subjecting the solution to a vacuum during pumping, a considerable economy is effected, both in the amount of zinc dust consumed, and in the acid required for the subsequent refining. Patent has been applied for in connection with this modification of the process.

Turbid and very cold solutions, extremely low in alkalinity and free cyanide, present the most unfavorable conditions for precipitation. The quantity of zinc dust required per ton is not proportional to the precious-metal content, depending largely on the amount used in side-reactions.

## Continuous Precipitation

In adding zinc dust to a "moving stream," as it is termed in the Merrill patent, a uniform feed is absolutely essential to secure maximum efficiency, the volume added per ton being often so small that any diminution becomes temporarily fatal to precipitation. Feeders of various forms have been designed and used. One of the first and most satisfactory consists of a slow-moving horizontal belt, on the level surface of which is spread a charge of zinc dust in a layer of uniform width and thickness. The zinc usually falls into a small mixing cone, through which an auxiliary stream of solution passes, carrying the zinc by a small pipe to the pump intake. A slow drip of lead acetate or nitrate, or of strong cyanide solution, may be added here to facilitate precipitation. Many ingenious elaborations have been devised to secure uniformity in the fall of zinc and flow of auxiliary solution and chemicals. Formerly

the zinc and auxiliary solution in the mixing cone were continuously agitated by a small jet of compressed air. According to data published by Clark[1] a saving of about one-third of the zinc was effected by eliminating this air agitation of the zinc feed, and preventing air from being drawn into the mixing cone.

Another good feeder has a cylindrical roller or pulley slowly revolving at the lower end of a hopper, conveying a narrow ribbon of dust, the thickness of which is controlled by an adjustable slot. Or an auger-like horizontal screw may remove the zinc from a similar hopper. All such hopper-fed devices require jarring mechanism to prevent the zinc dust from bridging. A miniature tube mill has been introduced as a mixer to smooth out irregularities in the zinc feed.

Bosqui's zinc-dust feed system has two relatively small tanks, alternately filled and emptied by means of a tilting launder which actuates a counter and throws a measured charge of zinc to the tank to be filled; the zinc is continuously stirred by a set of jets of solution on a revolving agitator.

Mills has a set of vacuum filter frames submerged in a vat through which the mixture of zinc and solution is circulated by a centrifugal pump, while settling is overcome by revolving rakes.

## Filter-pressing

Solution intended to be precipitated and thrown away may be run by gravity to a press at a low point; if for further use, it is generally pumped to a press at a considerable elevation, from which it falls to a storage tank. If the distance between intake and press is considered insufficient to allow of complete reaction, it is increased by leading the pipe in a zigzag line. A zinc press with gravity feed, the clear effluent elevated by a pump to storage, has the advantage over a pump-fed press in that zinc and precipitate are kept out of the cylinders and valve chambers; the claim is, however, made that the pump feed gives better precipitation with a given proportion of zinc.

A close filtering medium is necessary to retain zinc dust and the extremely fine precipitate obtained from low-grade gold solutions. In early practice the medium was sometimes paper between two cloths, or a single thickness of chain-cloth—a rather expensive fabric. Chain-cloth or cheaper heavy canvas may be covered with a light cheap twill, which is removed and burned at each cleanup to recover adhering precipitate, while the heavy backing is occasionally washed or treated with hydrochloric acid to remove limey accumulations. Another plan is to use two thicknesses of medium-weight cotton twill, the outer being taken off at

[1] *Mining Magazine* (April, 1910), **4**, 289.

each cleanup and either washed or burned. The other then becomes the outside cloth and a new or washed cloth is put under it for the next run.

One square foot of net filtering surface for 1.5 tons (say 50 cu. ft.) solution per day, or 6 tons per hour for 100 sq. ft., may be taken as a conservative ratio: with clear solutions a press may be run at double this rate for long periods. Colloidal suspended matter soon increases the pressure and reduces the pumping rate.

Two-inch distance-frames are suitable for a press used on gold solutions, 3-in. or 4-in. for silver. When a press is opened, most of the cake readily falls into the wheeled tray placed beneath and the remainder is removed by scrapers. It is normally soft but sometimes caked hard as a result of oxidation or the presence of calcium carbonate; this condition can often be controlled by excluding air from the solution.

It was early found advisable, and is still the custom, when starting up a zinc press after a cleanup, to add to the first charge 50 per cent. or more zinc in excess of the weight normally required; this is gradually diminished in successive charges, until the regular amount is reached. If the zinc feed is cut too low at any time, no effect may be noted until several charges have been thus treated, then the "barren" assays rise suddenly and it becomes necessary to add a considerable excess for several charges, until normal working is restored. Apparently a certain excess of zinc must be maintained in the press, or re-solution takes place there to some extent. This is evident when a press stands idle for several hours; the effluent samples caught during the first few minutes of pumping will be abnormally rich—occasionally richer than the pregnant solution entering the press.

## Fineness of Zinc Dust

The virtue of zinc dust as a precipitant is explicable by its fine state of division or, what amounts to the same thing, its extended surface. Waldstein's patent claimed that the zinc oxide, invariably present, formed a beneficial galvanic couple with the metal, while Sulman's process involved the preliminary removal of oxide by a solvent. In early practice it was noted that dust containing 1 or 2 per cent. of lead was more effective than purer samples, and this was confirmed by laboratory tests on synthetic alloys. Cadmium seems to have but little effect. That fine division is the main factor is indicated by the fact that the finest unoxidized metallic zinc, made by grinding sifted filings, and levigating in absolute alcohol, can be made even more effective as a precipitant.

To get some idea of working conditions in using zinc dust on the large scale, the particles may be assumed to be equal spheres of a diameter which we may take as 0.0001 in. One pound of this assumed zinc dust

will then contain 7544 million particles and will expose 1650 sq. ft. of surface, which is much greater than the surface of an equal weight of shaving. In precipitating gold, the practical minimum of zinc dust is probably between 0.1 and 0.2 lb. per fluid ton of 32 cu. ft., or 50 to 100 parts per million of solution. One-tenth of a pound, uniformly distributed through a ton of solution, would give some 13,650 particles per cubic inch, spaced at an average distance of about one twenty-fourth inch. Doubling the weight of dust per ton reduces the average distance between particles by about 20 per cent. If we could substitute particles of half the diameter, the same weights of zinc dust per ton would give eight times as many particles per cubic inch, at half the distance apart. These considerations make evident the desirability of obtaining zinc dust in a very fine state of division, and the bad effect of stray coarse shot or agglomerated masses. In strong solutions, rich in silver, it is probable that the economic limit of fineness would soon be reached; with gold solutions, it seems unlikely.

## Efficiency of Precipitation

Zinc dust is sometimes valued by the percentage of zinc actually in the metallic state, estimated by the reducing effect on ferric sulphate or chromic acid; or the difference between total zinc and zinc as oxide may be taken as metallic. As a guide to its precipitating value this is insufficient, and the fineness, or the speed of reaction, must be considered. As a rule, a good zinc dust will nearly all pass a sieve of 200 meshes to the inch, and in some zinc dusts now obtainable very little over 1 per cent. is retained by a 300-mesh sieve. In either case, the coarser portion is merely accidental material—crystalline aggregates, a few shots, or foreign matter.

A laboratory test based on precipitating effect is carried out[3] by shaking half a gram of the dust with a solution containing an excess of double silver cyanide and a little free alkaline cyanide, and determining the silver precipitated. This is a valuable but not infallible guide in practice. Herz[4] has shown that, to obtain comparable results, extreme care is necessary in adjusting the free alkali and cyanide in the test solution.

A precipitating efficiency of 100 per cent. assumes pure zinc, 1 atom of which should precipitate 2 atoms of silver or its equivalent in gold or copper from the double cyanide. At this rate 1 unit weight of zinc

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[3] W. J. Sharwood: Zinc-dust Tests. *Journal of the Chemical, Metallurgical and Mining Society of South Africa* (February, 1912), **12**, 332; *Mining and Scientific Press* (May 11, 1912), **104**, 659.

[4] *Trans.* (1915), **52**, 139.

should precipitate 6.03 units of gold, 3.30 of silver, or 1.93 of copper, but in actual practice the results are much lower. Applied to commercial dust, the laboratory test usually shows an efficiency value of 30 to 60 per cent. Some extremely fine and pure "artificial" zinc dusts, prepared by re-distillation of spelter, have shown laboratory efficiencies of 75 per cent. or more, and pure electrolytic zinc powder is claimed to be equally good. Otherwise the efficiency is generally higher when lead is present to the extent of a per cent. or two. Distinctly coarse or granular zinc preparations generally show very low results.

Judged by the actual precious metal precipitated, the working efficiency on a large scale may be a mere fraction of 1 per cent. in the case of low-grade gold solutions. A considerable amount of zinc is always wasted in side-reactions, such as the evolution of hydrogen, reducing dissolved oxygen, or precipitating copper and lead. With rich silver solutions the efficiency may approach 50 per cent., especially if no attempt is made to recover the last traces of silver.

A practical example of mixed precipitation may be taken from the records of the Drumlummon tailing plant, covering three seasons, or 24 months of treatment of sandy tailing by leaching, the silver being largely in excess of the gold.

| | |
|---|---|
| Sand treated | 290,000 tons |
| Solution precipitated | 390,000 tons |
| | Lb. Av. |
| Precipitate obtained | 104,000 |
| Zinc dust used | 113,600 |
| Metallic zinc in dust | 102,240 |
| Zinc remaining in precipitate | 40,300 = 39.4 per cent. |
| Zinc dissolved | 61,940 = 60.6 per cent. |
| Gold in precipitate | 2,380 = 0.39 per cent. efficiency. |
| Silver in precipitate | 16,000 = 4.75 per cent. efficiency. |
| Copper in precipitate | 7,800 = 3.93 per cent. efficiency. |
| Total efficiency (Au, Ag, Cu)............ | 9 07 per cent. efficiency. |

Percentage efficiency is calculated by dividing the weight in pounds by the electrochemical equivalent (Zn 32.7, Au 197.2, Ag 107.88, Cu 63.5, and Pb 103.6, under these conditions) dividing the quotient by the number of equivalents of zinc used, and multiplying by 100. The metallic zinc in dust has been taken throughout at the approximate figure of 90 per cent.

Another instance may be taken from the published results[2] of a month's run (May, 1911) of Sand Plant No. 1 of the Homestake Mining Co., in which the silver is about 1 per cent. of the gold value. Some lead was added to the solution as nitrate and recovered in the precipitate.

The efficiency for gold was 1.18 per cent., for silver, 0.73 per cent., and for copper 0.19 per cent., in the "weak solution," making a total

[2] *Transactions, Institution of Mining and Metallurgy* (1912), **22**, 142, 149. Tables XV, XVI, XXI.

efficiency of 2.1 per cent. Similarly the total efficiency (Au, Ag, Cu) in the "low solution" was only 0.71 per cent. At the same time about 1 per cent. of the zinc was consumed in precipitating lead.

That such low efficiencies are tolerated in gold extraction is explained by the fact that, when "low solution" is going to be thrown away, it is obviously worth while to extract the last 2-c. worth of gold recoverable, if this can be done at the expense of 1 c. for zinc, making a fair allowance for refining cost. With zinc at 14.5 c. per pound, 1 oz. Troy costs 1 c. If this ounce is used to precipitate 0.001 oz. of gold (2 c.) from a ton of waste solution the practice is defensible, although the actual chemical efficiency attained is less than 0.02 per cent.

## Zinc Dust *vs.* Zinc Shavings

The practical efficiencies obtained with zinc dust have been, generally speaking, about the same as with zinc shavings, and the accumulation of zinc in the solutions is about the same; greater variations occur between two plants using the same process on different ores than between two plants using the different precipitants on similar ores.

The dust process involves a more expensive installation than zinc shavings, but has the advantage of greater compactness and cleanliness, and involves less labor in maintenance and cleaning up as well as less risk of theft. The periodical cleanup is absolute, while a holdover of several thousand dollars' worth of precious metal commonly occurs with zinc shavings, and makes it impossible to compare the actual with what is often called the "theoretical" recovery. After a destructive fire, precipitate in a filter-press has been found intact, while zinc boxes have entailed great difficulty in the attempts to recover their contents. At a gold plant, a press occupying a floor space of 5 ft. by 14 ft. can easily carry a month's accumulation of $40,000.

The comparative cost of the two systems at any time depends, of course, upon the wage scale and the relative prices of zinc dust and spelter.

## Composition and Treatment of Precipitate

Except in the absence of coarse fibers in the former, there is no essential difference in the composition of zinc-dust precipitate and that obtained with shavings. Both contain metallic gold and silver, usually co-precipitated as an alloy, sometimes amalgamated with mercury or alloyed with copper or with zinc itself—some zinc being apparently reprecipitated electrolytically with the precious metal. Any lead in the solution is also thrown down, while any lead and cadmium in the zinc dust remain with the undissolved zinc, which may form 20 to 70 per cent. of the dry weight of precipitate. Calcium carbonate, moisture, and fine

ore particles make up the total, with sometimes calcium sulphate or zinc ferrocyanide and basic cyanide. Lead, mercury, and silver may be present either as metals or sulphides, the latter condition resulting from sulphur compounds in the solution. Blowing air through the press to dry the precipitate causes rapid oxidation of zinc, and consequent heating. Dilute sulphuric acid, followed by thorough washing, removes most of the zinc and lime, and more or less copper, while lead, cadmium, mercury and silica remain with the precious metals. The mercury may then be removed by drying and heating in a retort with a little lime. Imperfect washing leaves zinc sulphate, and this, like calcium sulphate or sulphides, yields a matte in the subsequent refining. Hitherto no way has been found to utilize the zinc sulphate solution, impure and generally saturated with calcium sulphate, which is obtained in the acid treatment of precipitate.

Acid treatment, or lead refining and cupellation, or a combination of the two, is necessary before melting low-grade precipitate. Richer gold precipitate, and that from most silver ores, may be fused with a suitable flux, directly or after roasting, yielding fairly fine bullion.

# Experiments in the Recovery of Tungsten and Gold in the Murray District, Idaho*

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(St Louis Meeting, October, 1917)

THERE is a small area about Murray where bedded gold quartz veins occur in Prichard slate. The ore taken from the upper levels, during the early days, was free milling and was treated by stamp milling and amalgamation. There was likewise a considerable placer working. The report is current that tungsten (scheelite) was encountered, which interfered with the amalgamation in the stamp mills and which, in placering, collected in the riffles in the sluice boxes and clogged them. Consequently, in the quartz mines, the scheelite, when encountered, was rejected, being thrown into the waste, sometimes inside the mine and at other times outside on the dump. The free surface ores became worked out and as the ore in depth became pyritic and rebellious to amalgamation the mines closed down.

Last spring tungsten ore brought a high price, near $80 per unit, 60 per cent. $WO_3$ and up. This so stimulated the output of tungsten ore that old dumps were explored and waste stored in the mines was gone over. Tungsten was likewise mined from virgin ground. So the district produced quite heavily for a time.

The condition now is different. The price is not so high, about $20 per unit $WO_3$, which makes profits questionable when mining in new ground. The accumulations from early day mining have been exhausted and the output has greatly declined.

The Golden Chest mine was the heaviest producer of tungsten. The months of June and July were spent at this mine studying the problem. Since the mine had produced both gold and tungsten, we looked forward to the problem of recovering both from what might be called a gold-tungsten ore. A study of the Golden Chest vein showed this was not to be the case. While gold ore and tungsten ore come from the same vein, the tungsten ore is confined to a more or less disturbed or distorted portion

*This research was partly done at the Golden Chest mine, Murray, Idaho, during the summer months of 1916, and completed in the laboratory of the University of Idaho, Moscow, Idaho, during the winter of 1916–17. This paper was originally presented at a meeting of the Columbia Section, on Feb. 23, 1917.

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of the vein, carrying almost no gold and designated as the tungsten zone. Ore carrying appreciable gold comes from other portions of the vein and carries no tungsten (see Figs. 1 and 2).

We have not considered the treatment of the gold ore, tungsten-free, because its treatment is a different problem. The gold in it is not free-milling. It is pyritic and rebellious to amalgamation.

While the tungsten ore, as it occurs in the Golden Chest vein, carries no gold and is not pyritic, it may become mixed, during the mining operation, with pyritic material carrying some gold broken from adjacent ground. The scheelite occurs in the quartz in rich pockets connected by stringers. The scheelite is usually soft and granular. Several hundred pounds, $WO_3$ 60 per cent. and up, may be sacked out of one pocket in the stope, ready for shipment. At other times it is firm and hard and disseminated to some extent through the quartz, though usually much segregated.

Since the tungsten ore in the solid occurs in these rich segregations, a considerable portion is sacked in the stope and sent out to a cleaning floor, where it is easily brought up to shipping grade by hand-picking. In breaking down the ore, some fine tungsten becomes mixed with waste rock and some pyritic material carrying a small amount of gold. The tungsten is recovered from this mill feed by concentration, using a hand-jig. Since the hand-jig is operated first with an 8-mesh screen and subsequently with a 60-mesh screen, when sufficient feed for the 60-mesh screen has accumulated, we will designate the milling operation as being done by two hand-jigs; one 8-mesh jig and one 60-mesh jig.

The mill feed was crushed through a 4-mesh screen. This was fed to the 8-mesh hand-jig. The products on the sieve were clean tailings, some middlings which were reground and returned to the system, and concentrates. There was produced in the hutch a product which was the feed to the 60-mesh hand-jig. The products on the sieve of the 60-mesh hand-jig were also clean tailings, some middlings which were reground and returned to the system, and concentrates. There was produced in the hutch a product containing about 18 per cent. $WO_3$. This hutch product of the 60-mesh hand-jig was not treated but was allowed to accumulate. The experiment was made in order to determine the best method of recovering the scheelite from this product. The losses in the tailings rejected from the 8-mesh hand-jig and the 60-mesh hand-jig were made practically nil, consequently the entire milling loss occurred in the treatment of this hutch product from the 60-mesh hand-jig. There was considerable preliminary work done which is not here reported. Table 1 gives all details relative to the final test on 4076 grams of this hutch product of the 60-mesh hand-jig which was sized by sieves into six sizes. Each size was treated separately on a 2-ft. Wilfley table. Each table operation produced only two products, tailings and concentrates, no

middlings being made. Pyrite has a specific gravity of 5.0; scheelite a specific gravity of 5.7. There is not sufficient difference between the two to enable them to be separated by a gravity method, consequently both of these minerals went into the concentrates. The concentrates were almost free of quartz, yet averaged only 53.7 per cent. $WO_3$, due to the high pyrite content. It is to be noted that no trouble was experienced in operating the Wilfley table, treating through-200-mesh material. This Wilfley made nearly the same recovery as the Wilfley operating with coarse feed. This would not be the case were a sulphide mineral, like galena, being treated. The reason for this difference in favor of scheelite is that scheelite does not tend to float. The tungsten recovery in the concentrates (raw) from the total Wilfley feed was 91.6 per cent.

TABLE 1—PART 1

| | Feed | | | | | | | | | | Concentrates (Raw) | | | | | | | | | Tails | | | | | | | | | Unaccounted for | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Chemical Analysis | | | | | | | Chemical Analysis | | | | | | | | | Chemical Analysis | | | | | | | | | | | | | |
| Jigs (Coarse and Fine) and Wilfley (No. 1 to 6) | Mesh to linear inch | Weight – grams | Percent of total Wilfley Feed | Cumulative Percent of total Wilfley Feed | $WO_3$ – percent | Fe – percent | S – percent | Au – oz per ton | $WO_3$ – Weight_gr $\left[\frac{(\text{Col No.3})\times(\text{Col No 6})}{100}\right]$ | Au – Weight_mg $\left[\frac{(\text{Col No.3})\times(\text{Col No 9})}{29.166}\right]$ | Weight – grams | $WO_3$ – percent | Fe – percent | S – percent | Au – oz per ton | $WO_3$ – Weight_gr $\left[\frac{(\text{Col No.12})\times(\text{Col No.13})}{100}\right]$ | Au – Weight_mg $\left[\frac{(\text{Col No.12})\times(\text{Col No 16})}{29.166}\right]$ | $WO_3$ recovered in Conc. – percent of $WO_3$ in Feed $\left[\frac{(\text{Col No 17})\times 100}{(\text{Col No 10})}\right]$ | Au recovered in Conc. – percent of Au in Feed $\left[\frac{(\text{Col No.18})\times 100}{(\text{Col No 11})}\right]$ | Weight – grams | $WO_3$ – percent | Fe – percent | S – percent | Au – oz per ton | $WO_3$ – Weight_gr $\left[\frac{(\text{Col No.21})\times(\text{Col No.22})}{100}\right]$ | Au – Weight_mg $\left[\frac{(\text{Col No 21})\times(\text{Col No 25})}{29.166}\right]$ | $WO_3$ lost in Tails – percent of $WO_3$ in Feed $\left[\frac{(\text{Col No.26})\times 100}{(\text{Col No.10})}\right]$ | Au lost in Tails – percent of Au in Feed $\left[\frac{(\text{Col No 27})\times 100}{\text{Col No 11}}\right]$ | Weight – grams [(Col No 3)–(Col No 12 + Col No 21)] | $WO_3$ – Weight – grams [(Col No.10)–(Col No 17 + Col No 26)] | Au – Weight – mg [(Col No 11)–(Col No.18 + Col No. 27)] | $WO_3$ unaccounted for – percent of $WO_3$ in Feed $\left[\frac{(\text{Col No 31})\times 100}{(\text{Col No. 10})}\right]$ | Au unaccounted for – percent of Au in Feed $\left[\frac{(\text{Col No.32})\times 100}{\text{Col No 11}}\right]$ | Ratio of Concentration – (tons into one) $\left[\frac{(\text{Col No 3})}{(\text{Col No 12})}\right]$ |
| Col No. 1 | No. 2 | No. 3 | No. 4 | No. 5 | No. 6 | No. 7 | No. 8 | No. 9 | No. 10 | No. 11 | No. 12 | No. 13 | No. 14 | No. 15 | No. 16 | No. 17 | No. 18 | No. 19 | No. 20 | No. 21 | No. 22 | No. 23 | No. 24 | No. 25 | No. 26 | No. 27 | No. 28 | No. 29 | No. 30 | No. 31 | No. 32 | No. 33 | No. 34 | No. 35 |
| Coarse Jig | 4–8 | | | | | | | | | | | 66.2 | 8.3 | 9.5 | | | | | | | | | | | | | | | | | | | | |
| Fine Jig | 8–60 | | | | | | | | | | | 46.3 | 17.7 | 19.7 | 0.665 | | | | | | | | | | | | | | | | | | | |
| Wilfley No. 1 | 60–80 | 489 | 12.0 | | 29.7 | 6.7 | 6.6 | 0.190 | 145.0 | 3.19 | 233 | 57.9 | 11.3 | 12.7 | 0.550 | 135.0 | 4.24 | 93.1 | | 246 | 0.7 | 1.8 | 0.36 | 1000 | 1.72 | 0.84 | 1.19 | | 10 | 8.28 | | 5.71 | | 2.10 |
| Wilfley No. 2 | 80–100 | 824 | 20.2 | 32.2 | 20.0 | 5.3 | 5.0 | 0.120 | 164.8 | 3.39 | 284 | 55.8 | 12.7 | 14.1 | 0.446 | 158.8 | 4.35 | 96.4 | | 529 | 0.3 | 1.5 | 0.24 | 0075 | 1.59 | 0.13 | 0.94 | | 11 | 4.41 | | 2.66 | | 2.90 |
| Wilfley No. 3 | 100–120 | 718 | 17.6 | 49.8 | 17.6 | 5.2 | 4.6 | 0.170 | 126.6 | 4.20 | 184 | 53.6 | 13.0 | 13.8 | 0.620 | 98.6 | 3.91 | 93.6 | | 483 | 0.4 | 1.4 | 0.12 | 0075 | 1.93 | 0.12 | 1.38 | | 17 | 10.20 | | 5.02 | | 3.90 |
| Wilfley No. 4 | 120–150 | 433 | 10.6 | 60.4 | 17.5 | 5.1 | 4.5 | 0.170 | 75.8 | 2.53 | 176 | 51.6 | 12.7 | 13.4 | 0.495 | 90.8 | 2.95 | | | 291 | 0.3 | 1.4 | 0.06 | 0050 | 0.87 | 0.05 | | | | | | | | 2.46 |
| Wilfley No. 5 | 150–200 | 446 | 11.0 | 71.4 | 16.9 | 5.0 | 4.1 | 0.180 | 75.5 | 2.76 | 130 | 45.5 | 11.5 | 10.7 | 0.450 | 59.1 | 2.01 | 78.3 | | 288 | 0.8 | 1.5 | 0.14 | 0350 | 2.30 | 0.34 | 3.04 | | 28 | 14.10 | | 18.66 | | 3.44 |
| Wilfley No. 6 | Thru 200 | 1166 | 28.6 | 100.0 | 12.7 | 4.2 | 2.3 | 0.190 | 148.0 | 7.61 | 247 | 53.5 | 10.1 | 9.6 | 0.750 | 132.0 | 6.35 | 89.2 | | 875 | 1.6 | 2.1 | 0.34 | 0200 | 14.00 | 0.60 | 9.45 | | 44 | 2.00 | | 1.35 | | 4.72 |
| Tot. Wil. Feed | | 4076 | 100.0 | | 18.0 | 4.9 | 4.2 | 0.170 | 735.7 | 23.68 | 1254 | 53.7 | | | 0.554 | 674.3 | 23.85 | 91.6 | 100.7 | 2712 | 0.83 | | | 0224 | 22.41 | 2.08 | 3.08 | 8.80 | 110 | 38.99 | –2.25 | 5.32 | –9.50 | 3.25 |

Certain buyers of tungsten concentrates asked for a $WO_3$ content greater than 60 per cent. and for a sulphur content less than 1 per cent. The 8-mesh hand-jig concentrates invariably fulfilled the condition as to $WO_3$, but not the condition as to sulphur. The 60-mesh hand-jig concentrates invariably had low $WO_3$ and high sulphur contents. All the Wilfley concentrates had low $WO_3$ and high sulphur contents. Two methods presented themselves for purifying the concentrates by removing the pyrite:

(*a*) Flotation.

(*b*) Roasting for magnetism followed by magnetic separation.

## FLOTATION[1]

The underflow (purified tungsten product) gave $WO_3$, 70.9 per cent.; iron, 2.5 per cent.; sulphur, 2.7 per cent. The overflow gave $WO_3$, 11.8

[1] Only one test was made. By further experimenting better results might be secured.

per cent.; iron, 36.7 per cent.; sulphur 41.0 per cent. The purified tungsten product was not bad, yet a much cleaner product was desired. The overflow carried considerable tungsten, which was a loss and, moreover, the pyritic tungsten concentrates, before they could be fed to the flotation machine, were required to be reground. The objections to this method are:

1. The purified tungsten concentrates are not sufficiently clean.
2. There is considerable loss of tungsten in the overflow.
3. Fine grinding is required.

## Roasting Followed by Magnetic Separation

Ten grams of pyritic tungsten concentrates were roasted in an assayer's gasoline-fired muffle furnace in a 2½-in. roasting dish. The

TABLE 1—PART 2

10 grams of Concentrates (Raw) roasted and separated by magnet into Purified $CaWO_4$ Concentrates and Magnetic Iron Sulphide

Purified $CaWO_4$ Concentrates

| Temperature of Roasting—degrees Centigrade | Duration of Roasting—minutes | Flaming began after being in Furnace—minutes—seconds | Flaming ceased after being in Furnace—minutes—seconds | Weight—grams [Roast that was made uncovered] | Weight—grams [Roast that was made covered] | Chemical Analysis: $WO_3$—percent | Chemical Analysis: Fe—percent | Chemical Analysis: S—percent | Chemical Analysis: Au—oz. per ton $\left[\frac{(\text{Col No 47}) \times 29.166}{(\text{Col No 41})}\right]$ | $WO_3$—weight gr $\left[\frac{(\text{Col No 41}) \times (\text{Col No 42})}{100}\right]$ | Au—weight—mg | $WO_3$ recovered in Purified $CaWO_4$ Conc.—percent of $WO_3$ in Conc (Raw) $\left[\frac{(\text{Col No 46}) \times 1000}{(\text{Col No 13})}\right]$ | $WO_3$ recovered in Purified $CaWO_4$ Conc.—percent of $WO_3$ in Feed $\left[\frac{(\text{Col No 48}) \times (\text{Col No 19})}{100}\right]$ | Au contained in Purified $CaWO_4$ Conc.—percent of Au in Conc (Raw) $\left[\frac{(\text{Col No 47}) \times 100}{(\text{Col No 52})}\right]$ | Au contained in Purified $CaWO_4$ Conc—percent of Au in Feed $\left[\frac{(\text{Col No 50}) \times (\text{Col No 20})}{100}\right]$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No. 36 | No. 37 | No. 38 | No. 39 | No. 40 | No. 41 | No. 42 | No. 43 | No. 44 | No. 45 | No. 46 | No. 47 | No. 48 | No. 49 | No. 50 | No. 51 |
| 850 | 5 | 2 | 4 | 7616 | | 78.4 | 0 30 | 0 08 | | 5.96 | | 90 0 | | | |
| 850 | 7 | 2 | 5 | 4.177 | 5.658 | 76 1 | 1 50 | 0.14 | 0 031 | 4.31 | 0.006 | 93.2 | | 2.63 | Same values as Col No 50 |
| 850 | 5 | 1–45 | 4 | 6.728 | 7355 | 79.5 | 0 33 | 0.08 | 0.226 | 5.85 | 0.057 | 101 0 | 94.1 | 31 30 | |
| 850 | 5 | 1–30 | 4 | 6.701 | 7.035 | 77.2 | 0 91 | 0 11 | 0.145 | 5.43 | 0.035 | 97.4 | 93 9 | 22.90 | |
| 850 | 5 | 1–15 | 3–50 | 6.670 | 6.897 | 74.5 | 1.15 | 0 16 | 0.283 | 5.15 | 0.067 | 96.1 | 90.0 | 31 50 | |
| 850 | 5 | 1–15 | 3–35 | 6.668 | 6.696 | 73 8 | 1 29 | 0.18 | 0 157 | 4.95 | 0.036 | 95 9 | 89.8 | 21 20 | |
| 850 | 5 | no flame | | 6.863 | | 64.8 | 1 67 | 0 15 | 0.208 | 4.45 | 0.049 | 97.8 | 76.6 | 31 60 | |
| 850 | 5 | no flame | | 7.216 | 7.228 | 68.9 | 3.20 | 0 12 | 0.500 | 4.98 | 0.124 | 93.1 | 83.8 | 48 10 | |
| | | | | | | | | | | | | | | | |

Magnetic Iron Sulphide; Unaccounted for

| Temperature of Roasting—degrees Centigrade | Au—wt mg contained in 10 gr of Conc (Raw) $\left[\frac{(\text{Col No 16}) \times 10}{29.166}\right]$ | Weight—grams—calculated assuming change $FeS_2$ to FeS $\left[10\,\text{gr} - \frac{\text{Col No 13}}{10 \times 0.806}\right] \times \frac{88}{120} + \left[\frac{\text{Col No 56}}{0.806}\right]$ | Chemical Analysis: $WO_3$—percent $\left[\frac{(\text{Col No 56}) \times 100}{(\text{Col No 53})}\right]$ | Chemical Analysis: Au—oz. per ton $\left[\frac{(\text{Col No 57}) \times 29.166}{(\text{Col No 53})}\right]$ | $WO_3$—weight—gr $\left[\frac{(\text{Col No 13})}{10} - (\text{Col No 46})\right]$ | Au—weight—mg | $WO_3$ contained in Mag Iron Sulphide—percent of $WO_3$ in Conc (Raw) $\left[\frac{(\text{Col No 56}) \times 1000}{(\text{Col No 13})}\right]$ | $WO_3$ contained in Mag Iron Sulphide—percent of $WO_3$ in Feed $\left[\frac{(\text{Col No 58}) \times (\text{Col No 19})}{100}\right]$ | Au recovered in Mag Iron Sulphide—percent of Au in Conc (Raw) $\left[\frac{(\text{Col No 57}) \times 100}{\text{Col No 52}}\right]$ | Au recovered in Mag Iron Sulphide—percent of Au in Feed $\left[\frac{(\text{Col No 60}) \times (\text{Col No 20})}{100}\right]$ | Unaccounted for: Au—weight—mg $[(\text{Col No 52}) - (\text{Col No 47} + \text{Col No 57})]$ | Unaccounted for: Au unaccounted for—percent of Au in Conc (Raw) $\left[\frac{(\text{Col No 62}) \times 100}{(\text{Col No 52})}\right]$ | Unaccounted for: Au unaccounted for—percent of Au in Feed $\left[\frac{(\text{Col No 65}) \times (\text{Col No 20})}{100}\right]$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No. 36 | No. 52 | No. 53 | No. 54 | No. 55 | No. 56 | No. 57 | No. 58 | No 59 | No 60 | No 61 | No 62 | No 63 | Col No 64 |
| 850 | | | | | | | | | | | | | |
| 850 | 0.228 | 3.53 | | 2.25 | 0 32 | 0.272 | 6 80 | | 119 2 | Same values as Col No 60 | −0.056 | −21 83 | Same values as Col No 63 |
| 850 | 0.182 | 2.06 | −2.91 | 1 75 | −0.06 | 0.124 | −1 00 | −1 0 | 68 2 | | 0.001 | 0.50 | |
| 850 | 0.153 | 2.45 | 6.12 | 1 70 | 0.15 | 0.143 | 2.6 | 2.5 | 93.5 | | −0.025 | −16 40 | |
| 850 | 0.213 | 2 72 | 7 72 | 1.61 | 0.21 | 0.150 | 3 9 | 3 6 | 70.4 | | −0 004 | −1 90 | |
| 850 | 0.170 | 2 89 | 7.28 | 1 18 | 0 21 | 0 117 | 4.1 | 3.8 | 68.8 | | 0 017 | 10 00 | |
| 850 | 0.155 | 3.30 | 3.03 | 0.94 | 0 10 | 0.106 | 2.2 | 1 7 | 68 4 | | 0 000 | 0 00 | |
| 850 | 0.258 | 2 93 | 12.60 | 1 32 | 0 37 | 0.132 | 6 9 | 8 1 | 51 2 | | 0 002 | 0 70 | |
| | | | | | | | | | | | | | |

temperature was measured by a thermo-electric pyrometer. Preliminary roasts to determine the best roasting temperature were made at 700°, 800°, 850°, and 900° C. The temperature of 850° C. gave the best results and this temperature was used in making the final complete test. The roasting dish was not covered and the roast was stirred occasionally. Flaming ceased slightly before the completion of the roast. It was believed that this method of roasting would produce conditions similar to those that would exist in practice when treating the material in quantity.

In the final test, two series of roasts were made. The temperature of 850° C. was used in both series. The first series was made uncovered and the material was stirred. The magnetic separation of the roasted material was made, producing purified $CaWO_4$ concentrates and magnetic iron sulphide. The purified $CaWO_4$ concentrates were weighed, then the en-

tire amount was ground fine and used in making the determinations of $WO_3$, iron, and sulphur. Gold values were desired in the purified $CaWO_4$ concentrates and in the magnetic iron sulphide. These values had not been determined and the materials were used up. Consequently, it was necessary to make a second series of roasts in order to provide materials for use in the determination of the gold. Since it had been noted that in making the roasts there was some loss of material by decrepitation, especially with the coarse sizes, it was decided to make this second series of roasts closely covered and not to stir the material at all.

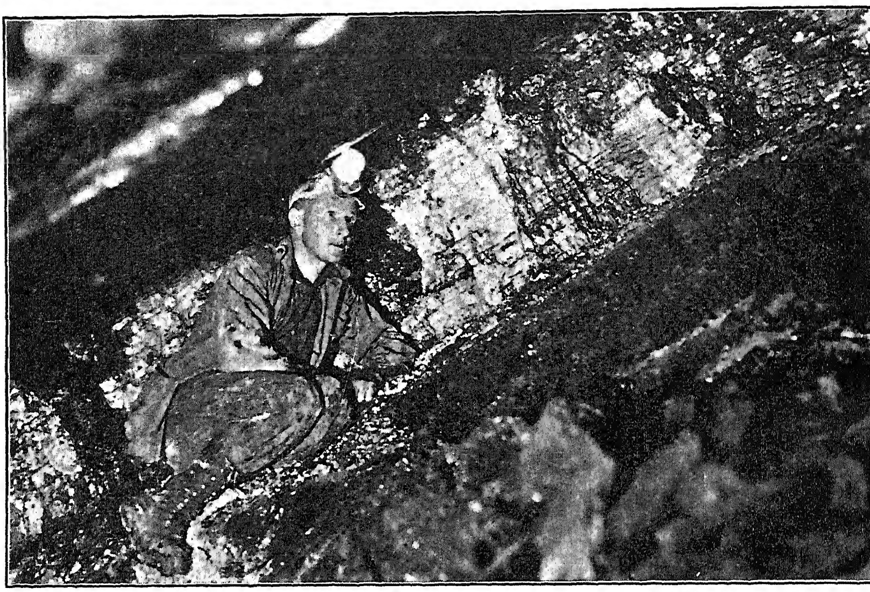

FIG. 1.—VEIN NOT IN TUNGSTEN ZONE, GOLD BEARING.

The second series of roasts was made closely covered and the decrepitation loss was thus entirely eliminated, facilitating the accounting for material. The roasts were not uncovered until they were out of the furnace and cold. The results of roasting covered were very satisfactory. Magnetic separation of the roasted material was made in the same way as in the first series of roasts, producing purified $CaWO_4$ concentrates and magnetic iron sulphide. The purified $CaWO_4$ concentrates produced, when roasting had been carried on uncovered (first series of roasts), had a slight reddish tinge due to the formation of some $Fe_2O_3$ that remained with and discolored the $CaWO_4$ which would otherwise have been white. When the roast that had been made covered (second series) was subjected to magnetic separation, the magnetic iron sulphide appeared more strongly

magnetic than that produced when roasting uncovered. The result was that the magnetic separation of this second series of roasts was accomplished with greater ease than that of the first series. The resultant purified $CaWO_4$ concentrates were white and had no reddish discoloration as did the product from the first series of roasts. The two products of this second series of roasts, the purified $CaWO_4$ concentrates and the magnetic iron sulphide, were then used for determining the gold content of each.

On examination under the microscope, the purified $CaWO_4$ concentrates resulting from the second series of roasts appeared as clean as the product of the first series. If there was any difference, the product of the

FIG. 2.—VEIN IN THE TUNGSTEN ZONE. POCKET OF TUNGSTEN (SCHEELITE) MARKED S.

second series appeared slightly cleaner. The weight of the purified $CaWO_4$ concentrates, resulting from the covered roast, was used as a basis of balancing quantities. The gold was likewise determined in the products of the second series of roasts, while the $WO_3$, iron, and sulphur were determined in the products of the first series. Decrepitation loss was thus eliminated. It is believed that if there is any error introduced by thus combining the data obtained from the two series of roasts, it is very small and on the safe side, for if determinations of the $WO_3$, iron, and sulphur had been made on the products of the second series of roasts, the $WO_3$ would probably have been slightly higher and the iron and the sul-

phur slightly lower than the values reported. It is to be noted that on the finest size, through-200-mesh, where the decrepitation loss was known to be practically nil when roasting uncovered, there is practically no difference in the weights of the purified $CaWO_4$ concentrates from the two series of roasts.

In making the magnetic separation, two magnets were used:

(a) Roughing magnet.[2]

(b) Finishing magnet.

The roughing magnet removed easily the bulk of the magnetic material. After using the roughing magnet, there remained a small amount of weakly magnetic material which required for removal a stronger magnet.

The purified $CaWO_4$ concentrates have a $WO_3$ content of 64 to 78 per cent., which is not far below that of the pure mineral, 80.6 per cent. The recovery of $WO_3$ in the purified $CaWO_4$ concentrates, in all but the through-200-mesh product, is greater than 95 per cent. of the $WO_3$ in the concentrates (raw). In purifying the through-200-mesh size, there was a tendency for more $CaWO_4$ to be entangled in the magnetic material and to be carried with it, and in treating that material, there was recovery in the purified $CaWO_4$ concentrates only 93.1 per cent. of the $WO_3$ in the concentrates (raw). There is thus but a small loss of $WO_3$ in the magnetic iron sulphide. The iron and sulphur are both low in the purified $CaWO_4$ concentrates. The iron in all but the two finest sizes is around 1 per cent., and the sulphur about 0.1 per cent. in all sizes.

The net recovery of $WO_3$ in the double operation, gravity concentration and magnetic separation, is the product of the two separate recoveries. This is recorded in the column of the tabulation headed, "$WO_3$ recovered in purified $CaWO_4$ concentrates, per cent. of $WO_3$ in feed." On all but the two finest sizes, this net recovery is practically 90 per cent. or greater, and on the two finest sizes 76 to 83 per cent.

## The Saving of Gold

As was stated at the start, we are not treating a gold ore in this experiment. The material treated is a tungsten material containing a small amount of gold. The distribution of the gold in the different products of this experiment has been investigated in order to make the experiment complete. The magnetic iron sulphide separated, although carrying an ounce of gold per ton, is of small importance commercially because of its small tonnage. For instance, suppose 4 tons of purified $CaWO_4$ concentrates were produced in a month. There would be produced less than 2 tons of magnetic iron sulphide carrying about 1 oz. of gold per ton and of a gross value of about $40. Consequently the investigation of the

[2] Separating magnet, manufactured and sold by A. H. Kidney, 255 Park Avenue, Orange, N. J.

distribution of the gold in the products is interesting only from a technical standpoint. The profits must be derived from the output of purified $CaWO_4$ concentrates.

## SUMMARY

1. In the Golden Chest vein, the tungsten mineral, scheelite, occurs in rich segregations, in a more or less disturbed or distorted portion of the vein known as the tungsten zone, barren of gold and pyrite-free.

2. Since the tungsten mineral in the vein occurs in these rich segregations, a considerable portion of the ore is sacked in the stope and sent out to a cleaning floor, where it is brought up to a shipping grade by hand-picking.

3. In breaking down the ore in the mine, some fine tungsten mineral becomes mixed with waste rock and some pyritic material carrying small gold values. The tungsten mineral is recovered from this material (mill feed) by concentration.

4. Gravity concentration produces pyritic tungsten concentrates, 53.7 per cent. $WO_3$, with high pyrite content. This is not a shipping grade.

5. The objections to flotation, as a means of purifying the pyritic tungsten concentrates, are: The purified tungsten concentrates are not sufficiently clean; there is considerable loss of tungsten in the overflow; and fine grinding is required.

6. The pyritic tungsten concentrates, when treated by roasting for magnetism followed by magnetic separation, yield purified $CaWO_4$ concentrates of shipping grade, $WO_3$ greater than 60 per cent., and sulphur less than 1 per cent. On all but the two finest sizes, the net recovery, expressed as per cent. of $WO_3$ in the mill feed, is practically 90 per cent. or greater, and on the two finest sizes, 76 to 83 per cent.

7. The investigation of the distribution of gold in the products is interesting only from a technical and not from a commercial standpoint. The profits must be derived from the output of purified $CaWO_4$ concentrates.

# Exploration of Metalliferous Deposits

BY W. H. EMMONS,* PH. D., MINNEAPOLIS, MINN.

(St. Louis Meeting, September, 1917)

## Introduction

THE exploration of deposits of the metals will never become an exact science. There will always be an element of uncertainty in prospecting and developing mines. In countries where the surface has been closely scrutinized, most of the deposits whose outcrops contain valuable metals have probably been discovered. Many metalliferous deposits, however, are normally so much altered at the surface that the true significance of their outcrops is likely to be obscured.

If a deposit is not exposed, its presence may become known by some one of its characteristics that is different in kind or in degree from a similar feature of the associated rock. Thus, a deposit that is not exposed may be discovered because of its magnetic property. So few minerals are strongly magnetic that the value of magnetic surveys is limited, but the dip needle has nevertheless proved of great aid in locating magnetic belts, and these in some regions are associated with workable iron ores. The Cuyuna Iron Range of Minnesota, which does not exhibit any outcrop of iron ore, was discovered by drilling "areas of attraction" that are due to belts of magnetic rocks associated with the iron ores. In some regions the use of the dip needle is justified, even if it does no more than reveal the strike of the rocks, since a knowledge of the latter enables the driller to locate and point his holes to advantage.

The measures of gravity by the use of the pendulum are so accurate, or the limit of error is so small, that one might detect the presence of a concealed tabular deposit approximately at the surface, 34 ft. (10.36 m.) thick, having a density double that of the surrounding rocks. However, not many deposits have a density twice that of ordinary rocks and a thickness as great as 34 ft.; if they lie a short distance below the surface the density or the thickness must be even greater. Consequently, there is not much hope of sufficiently developing the gravity method of detecting deposits of heavy ores to make it of practical value. If the limit of error could be reduced to about one-tenth the present factor, and if the rather tedious and painstaking methods of determining gravity that are now in use could be simplified, the method would promise some degree

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of usefulness in the exploration of certain types of heavy ores. This, however, seems unlikely, since great refinement of method is necessary to determine gravity with the degree of error that now exists. In the future, as in the past, the explorer will probably have to rely upon what he sees rather than upon some peculiar characteristic of concealed deposits.

Certain relations of the distribution of lode ores to geologic structures have long been recognized. These ores are deposited principally in regions of complex faulting and fracturing and in areas of igneous activity. The earlier prospector found that it was profitable to scrutinize the mountainous regions, for these in general are most complexly faulted and fractured, and are more likely to have been the centers of great igneous activities. In some regions, it is true, flat-lying rocks far removed from igneous centers yield deposits of the valuable metals; these, however, are rarely of the lode type.

Certain features of a region other than its outcrops may lead one to suspect the presence of ore lodes. Thus, the presence of placers will lead to a search for the sources of the metals they contain. In some regions, waters containing iron sulphate suggest the presence of deposits containing iron sulphides. At some places, many feet above the present streams, gravels are cemented by iron oxides that were evidently derived from the weathering of deposits containing iron sulphides. At Cananea, Mex., at Bingham, Utah, at Lead, S. D., below the Homestake lode, and in many other regions, iron-cemented gravels are conspicuously developed. These gravels were consolidated, long before the mines above them were opened, doubtless by iron oxide that was deposited by hydrolysis of iron sulphate formed by oxidation of pyrite during the weathering of the pyritic deposits.

Lode ores, it is believed, are deposited principally by ascending hot waters. This inference is justified since they are almost universally associated with intrusive igneous rocks. The solutions that course through fractures and shattered zones soak into the country rock. In some districts these solutions have wandered far from the master fractures, profoundly altering great areas of country rock. Where the prevailing hydrothermal alterations are sericitic, as at Butte, Mont., and in many other regions, the country rock is bleached. Where the alterations are propylitic, as in many of the precious-metal deposits of Nevada, igneous rocks by development of chlorites from dark minerals, become pale green. Prospectors, even those without academic training, know the significance of hydrothermal metamorphism. Some designate the result of such alteration as the "kindly look" of the rock and contrast it with the "hungry look" of the fresh unaltered rock which they have often found to be barren of ores.

Primary ore deposits fall into a few well-defined groups, each with characteristic features. These in different districts have definite rela-

tions to the geologic structure depending on their genesis. The mapping of the structures and the investigation of the relations of the deposits to structures are essential features of rational exploration.

Of the discoveries made in the United States within the past 10 years, perhaps as many have resulted from a knowledge of the geologic conditions of the districts containing the deposits as from the exploration of outcrops that were supposed to be the altered cappings of valuable deposits. The exploration made with an adequate knowledge of the structure of an area and also with an understanding of the characteristic superficial alteration of its deposits is the most likely to succeed.

Some outcrops themselves contain the valuable metals; others have had the valuable metals leached from them, but may carry the alteration products of associated minerals and these frequently furnish the clue leading to the discovery of orebodies. Pyritic copper sulphide deposits and pyritic zinc sulphide deposits are generally leached of copper and zinc near the surface. In the earlier days of mining many of the deposits were discovered incidental to mining associated metals less readily dissolved at the surface. In general these deposits were stained with iron oxide. The copper deposits and zinc deposits of Butte, Mont., were discovered by exploration of silver ores in the upper oxidized parts of the veins. The United Verde mine, Jerome, Ariz., the Highland Boy mine, Bingham, Utah, and the Mount Morgan mine, Queensland, were worked first as gold mines. All subsequently developed great bodies of copper ores carrying noteworthy amounts of gold.

Stimulated by examples of iron-stained rocks passing downward into workable copper ores, explorers investigated ferruginous outcrops systematically and many deposits were discovered below the valueless cappings. Many of the disseminated copper deposits are barren of copper at the surface. The exploration of these deposits soon showed, also, that some are not everywhere capped by gossans heavily stained with iron oxide. The cap rock in general is composed largely of sericite, kaolin, and chalcedonic silica. Some contain also sulphates. Such associations, though not so easily recognized as limonitic areas, are nevertheless significant and the recognition of the origin of such an outcrop is certain to play an increasingly important part in the explorations of the future. Because such an association of alteration products is less readily recognized, outcrops of this character are the most likely to have been overlooked in an area that has not been fully explored.

The problems connected with the superficial alteration of ore deposits, although complex, are susceptible of analysis and experimental study. The changes are accomplished by water, air, and the compounds resulting from the action of water and air upon the ore itself. One may take the ore, expose it to water and air, and ascertain what products are formed. One may expose the ore further to the action of water and the

products of alteration in the absence of air, and ascertain the changes that take place. By analysis one may ascertain the composition of the waters of mines. There are now available more than 50 analyses of waters from mines of sulphide ores.[1] These are similar in composition and closely resemble solutions formed by placing sulphide ore in contact with pure water. It may be assumed with confidence that solutions that have accomplished superficial alteration of sulphide ores are systems of sulphates, carbonates, and chlorides, of heavy metals, of alkalies and of alkaline earths.

Near the surface solutions are acid and generally contain ferric sulphate. In depth, ferric sulphate is reduced to ferrous sulphate, acidity is decreased by reaction with minerals of the ore and wall rock, and at greater depths the solutions become neutral and ultimately alkaline. Subjecting for long periods various minerals and various combinations of minerals to solutions such as are known to accomplish superficial alteration, one may ascertain what changes take place and compare the reactivities of ores of the various metals and their reactivities in various mineral associations.[1]

Some of the metals are easily dissolved near the surface where waters are acid and oxidizing. Of these some are precipitated in depth where acidity decreases and where solutions are reduced because oxygen of the air is excluded. Deposits of such metals are likely to be leached at the surface. If the metals are readily precipitated in depth the deposits become enriched below the surface. Other metals are very difficultly soluble and because associated materials are dissolved and removed deposits of such metals are likely to be enriched at the surface. Associated gangue minerals are important also because some metals dissolve readily in certain associations but not in others. One may separate the metals into groups—one of metals that dissolve readily, another of metals that dissolve slowly, and still another of metals that dissolve very slowly.

| 1. Dissolve Readily | 2. Dissolve Slowly | 3. Dissolve Very Slowly |
|---|---|---|
| copper | mercury | gold (in part) |
| zinc | lead | bismuth |
| silver | antimony | tin |
| gold (in part) | arsenic | chromium |
| uranium | | molybdenum |
| vanadium | | tungsten |
| iron | | |
| manganese | | |
| nickel | | |
| cobalt | | |

## Copper

Copper, because of its chemical relations, is easily leached from the surface and precipitated in depth. As already observed, the outcrops

[1] W. H. Emmons: The Enrichment of Ore Deposits, *U. S. Geological Survey, Bulletin* 625 (1917).

of a copper sulphide deposit may be so thoroughly leached that practically all the copper is removed and carried downward.

In base-leveled countries or in countries where the surface has remained nearly stationary for a long time, the outcrops are generally depleted of copper. Even in mountainous countries, where erosion is comparatively rapid, not many large deposits of copper are workable at the surface. Ferric sulphate hydrolyzes, depositing limonite, so the deposits of many iron-copper sulphide ores are marked by a gossan or "iron hat." As already noted, many copper deposits have been discovered by following downward a nearly barren gossan or by the downward exploitation of deposits of precious metals that are concentrated near the surface above deposits of copper ores in which the precious metals are present in small amounts.

In copper deposits that do not carry sulphides the downward transportation of copper is generally slow. The native copper deposits of Keweenaw Point, Mich., are workable at the surface, although the country has undergone erosion for a period so long that it has become nearly a peneplain.

Where the sulphides are present in subordinate quantities, copper carbonates and silicates may occur abundantly at and near the surface, as at Ajo, Ariz.,[2] where oxidized copper minerals are conspicuous in outcrops.

In limestone, copper will commonly segregate as carbonate at and near the surface, and many oxidized copper deposits in limestone have been worked by open pits. Limestones that have been altered by contact metamorphism are relatively impermeable, because their tough, heavy silicates, such as garnet, amphibole, and mica, are not readily fractured. Most deposits of this nature contain considerable calcite, and any copper-iron sulphide ore they carry will usually oxidize to carbonates, silicates, and oxides. The copper in such an ore is particularly stable and is likely to endure long weathering. Such deposits have stimulated deep prospecting in many districts where other types of deeper copper ores are present,[3] and they have thus served as useful indicators of hidden wealth.

The copper lodes at Butte (in igneous rocks) are leached of copper, some of them to a depth of 400 ft. (121.9 m.). The disseminated ores in porphyry show great variation as to depth of leaching, but are commonly leached to depths of 100 to 300 ft. (30.5 to 91.4 m.) below the surface, and exceptionally to greater depths. Some of them show practically no copper at the surface. At Cananea, Sonora, and Morenci, Ariz., barren gossans that were explored to considerable depths have led to good

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[2] I. B. Joralemon: The Ajo Copper-Mining District, *Trans.* (1915), **49**, 601.

[3] W. H. Emmons: The Outcrops of Ore Deposits, in *Types of Ore Deposits*, H. F. Bain and others, 318. San Francisco, 1911.

deposits of chalcocite ore. Copper was only sparingly present at most places in the outcrops of the great disseminated deposits at Miami and Ray, Ariz. At Bingham it was locally somewhat conspicuous as carbonates and silicates.

In making explorations for copper the question frequently is raised whether drilling is justified in an area that shows but little iron oxide at the surface. Nearly all copper deposits mined in North America do show ferruginous outcrops, but some gossans that cap valuable disseminated ores in porphyry are not heavily stained with iron. At Cananea, Sonora, valuable chalcocite deposits occur below outcrops that show heavy iron stain only here and there. As stated above, however, the outcrops show much silicification and kaolinization, and generally some limonite is present.

As a rule, the disseminated deposits of chalcocite ore will show outcrops more highly stained with iron in the earlier stages of their chalcocitization. As the country is eroded and the chalcocite zone descends, more and more pyrite and chalcopyrite are replaced by copper sulphides. A point may be reached where the chalcocite ore contains very little of the original iron sulphide. Obviously the oxidation of such an ore would yield but little iron sulphate. It would yield even less limonite, if the solutions were actively descending. If the process goes still further, the proportion of iron may become insufficient to dissolve all the copper in the chalcocite zone. The gossan then will generally carry oxidized copper minerals such as cuprite, the carbonates, basic sulphates, and chrysocolla.

## Zinc

Zinc, like copper, is very readily dissolved from its sulphide deposits and in the presence of much pyrite or other iron sulphide, zinc near the surface is likely to be almost completely removed from its deposits. It forms the very soluble zinc sulphate. But zinc in depth is not so readily precipitated by sulphides as copper. It will remain dissolved in solutions from which copper may be precipitated. Thus secondary concentrations of zinc sulphide ores are rare compared with the secondary ores of copper.

In limestone, a solution of zinc sulphate precipitates zinc carbonate ores. These ores have lately become prominent, particularly in several districts of western America, where superficial changes have produced notable concentration of zinc.[4]

[4] G. M. Butler: Some Recent Developments at Leadville, *Economic Geology* (1913), **8**, 1.

Adolph Knopf: Mineral Resources of the Inyo and White Mountains, Cal., *U. S. Geological Survey, Bulletin* 540, (1914) 97.

G. F. Loughlin: The Oxidized Zinc Ores of Tintic District, Utah, *Economic Geology* (1914), **9**, 1.

A common type of ore in limestone consists of pyrite, argentiferous galena, sphalerite, a little chalcopyrite, and other sulphides in a gangue of quartz. The orebodies, like many deposits in limestone, are commonly large irregular masses. In the oxidation of such a deposit the lead and much of the silver remain essentially in place, the galena being in part oxidized to anglesite and cerusite. The oxidation of pyrite and sphalerite yields acid, and zinc and iron sulphates, which are carried out of the deposits in great quantities. A part of the iron remains behind as oxide, but in some deposits practically all the zinc is removed. When the solution, which is doubtless acid and carries ferric, ferrous, and zinc sulphates, moving along a water channel, encounters the limestone that surrounds the orebody, it will precipitate iron and zinc.

Under some conditions a zinc-iron carbonate or sideritic smithsonite, monheimite, is formed. This reaction has recently been investigated by Wells.[5] Dilute solutions of two metallic salts in equivalent (molar) quantities were precipitated with only enough sodium carbonate for one metal. With equivalent quantities of zinc and calcium nearly all the zinc and only a trace of calcium are precipitated. With equivalent quantities of iron and calcium, nearly all the iron and only a trace of calcium are precipitated. Sideritic smithsonite, or monheimite, contains iron carbonate in varying proportions. Some smithsonite is nearly pure and some contains as much as 20 per cent. of iron carbonate, or even more.

After it is formed the smithsonite, or monheimite, with the progress of the erosion of the country, is exposed to more highly oxygenated waters. The iron carbonate then oxidizes and stains the ore brown so that it may easily be mistaken for iron-stained limestone. Thus deposits of this character, though exposed in underground workings, have been overlooked for years.

## Silver

Silver, like copper, is readily dissolved in ground waters. If the chloride forms in the gossan its solution is delayed. The chloride, however, is to be regarded as a temporary mineral except in arid countries[6] where chlorine is commonly present in considerable quantities in earth waters. In such countries more chloride forms and, moreover, its solution is prevented by chlorides present in ground waters. There may, therefore, be great enrichment of silver at and near the surface. In depth, silver is carried in acid solution. Air or ferric iron is necessary for its

[5] R. C. Wells: The Fractional Precipitation of Carbonates, *Washington Academy Scientific Journal* (1911), **1**, 21.

[6] R. A. F. Penrose, Jr.: The Superficial Alteration of Ore Deposits, *Journal of Geology* (1892), **2**, 288–317.

active solution in sulphate waters.[7] However, the carbonate of silver is soluble and without much doubt some silver is carried downward in carbonated waters. Silver is precipitated from its sulphate[8] and carbonate[9] solutions in a sulphide environment. For this reason great bodies of secondary silver ore may be deposited below a leached or low-grade capping.

## Gold

In acid waters that carry chloride in the presence of manganese oxides,[10] gold is readily dissolved; by reduction of acidity, gold is precipitated. Frequently the manganese and gold are precipitated together; in deposits that contain manganese, gold may be carried downward in solution and accumulate below the outcrop. But gold is readily precipitated from its solutions by many minerals, and its migration is slow. In the presence of carbonates,[11] or of other minerals that reduce acidity readily, gold in manganiferous deposits tends to remain near the surface and in manganiferous carbonate gangue it may even accumulate as placers. If the metal were not of great value, its secondary concentration would be of little economic importance because it is dissolved near the surface only in chloride solution and in solutions that reconcentrate sulphide ores and chlorides are much less abundant than sulphates. In deposits which do not contain manganese, gold dissolves very slowly, if at all, and tends to accumulate at the surface. Such deposits are commonly enriched by removal of material other than gold.

## Uranium and Vanadium

Uranium and vanadium are readily dissolved in sulphate waters and both are regarded as mobile metals. Both metals are precipitated from the soluble salts by organic matter.[12] The carnotite deposits of Colorado[13] are believed to be concentrations from cold ground waters that dissolved the metals from associated rocks. In many deposits, vanadium minerals occur in relations that leave no doubt of their secondary origin.

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[7] H. C. Cooke: The Secondary Enrichment of Silver Ores, *Journal of Geology* (1912), **21**, 9.

[8] Chase Palmer and E. S. Bastin: Metallic Minerals as Precipitants of Silver and Gold, *Economic Geology* (1913), **8**, 140.

F. F. Grout: *Idem.* (1913), **8**, 407–433.

[9] L. G. Ravicz: Enrichment of Silver Ores, *Economic Geology* (1915), **10**, 368–392.

[10] W. H. Emmons: The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold Deposits in the United States, *Trans.* (1913), **42**, 3–73.

[11] A. D. Brokaw: The Secondary Precipitation of Gold in Orebodies, *Journal of Geology* (1913), **21**, 251–268.

[12] F. L. Hess: An Hypothesis for the Origin of the Carnotites of Colorado and Utah, *Economic Geology* (1914), **9**, 675–688.

[13] W. F. Hillebrand and F. L. Ransome: On Carnotite and Associated Vanadiferous Minerals in Western Colorado, *U. S. Geological Survey, Bulletin* 262 (1905), 14.

## Iron and Manganese

Iron and manganese are grouped with the metals that are readily dissolved in sulphide ores. In sulphide deposits containing iron and manganese these metals are generally present in ground waters. The iron and manganese solutions, however, are not very stable and the oxides of iron and manganese are readily precipitated by hydrolysis. Consequently deposits that contain appreciable amounts of iron and manganese will generally carry these metals at the surface. Secondary deposits of iron sulphide or manganese sulphide are almost unknown. At depth, iron and manganese are deposited along fractures as oxides and more rarely as carbonates, but these ores in lode deposits are of relatively small importance. Deposits of iron and manganese are commonly enriched by removal of other materials. In general, iron and manganese deposits carry rich ores at the surface.

## Nickel

Nickel is similar to iron in its chemical activities, but unlike iron it does not oxidize in bivalent to form trivalent salts and its sulphate does not hydrolize readily and deposit oxide. It is dissolved almost as readily as copper in its sulphide combinations and it is precipitated in depth as sulphide. It is not so easily precipitated as copper sulphide, however, and will not be thrown down in an acid environment. It therefore resembles zinc in its migrational reactivities more closely than it resembles copper. No large deposits of secondary nickel sulphide have been recognized. The best-known deposits of nickel sulphide ores have been glaciated and possibly secondary sulphide zones have been removed. At the Lancaster Gap nickel mine,[14] Pennsylvania, secondary millerite has been present in bodies of economic value. The gossan of nickeliferous pyrrhotite is essentially limonite. In the weathering of nickeliferous basic rocks the nickel accumulates as silicates not far below the surface.

## Cobalt

Cobalt salts are soluble and cobalt is dissolved readily from outcrops. No deposits of secondary sulphides are known. Its chemical behavior is closely similar to that of nickel and zinc and it should be grouped with the metals which are concentrated under favorable conditions. Like nickel, cobalt forms silicates and concentrates in the superficial zones of silicate deposits which are undergoing alteration. Cobalt, like nickel, forms a moderately insoluble arsenate in some outcrops. Cobalt is found in considerable quantities in the oxidized material which caps the altered peridotite of New Caledonia.

[14] J. F. Kemp: The Lancaster Gap Nickel Mine, *Trans.*, (1894), 24, 620.

## Mercury

Mercury is dissolved and reprecipitated in underground waters. The reactions go on very readily with chloride waters but not in sulphate solutions. Mercuric chloride is soluble in water and does not give insoluble basic salts with water. Mercuric sulphate, on the other hand, is easily hydrolized and gives basic sulphate which is reduced to native metal. Experiments made over long periods by Broderick[15] show that cinnabar is practically insoluble in sulphate waters although it is readily dissolved in hydrochloric acid, and more readily still in presence of manganese oxide. The secondary enrichment of mercury deposits is somewhat similar to that of gold deposits. Neither metal is appreciably dissolved except in presence of chlorides. Reconcentration of mercury is most marked in arid countries, where chlorides are present. Chlorine, however, is quite subordinate in underground waters compared with sulphates. Underground waters, therefore, can not move large masses of the metal and concentrate them as copper and silver are concentrated. Gold is so valuable that even small concentrations are important economically. Mercury is less valuable: a concentration of 2 oz. of gold per ton would be highly significant, but 2 oz. of mercury per ton would be unimportant. There is indisputable evidence that secondary mercury minerals result from processes of sulphide enrichment, but these are small in amount because chlorides in general are not abundant in waters of deposits of mercury ores. Mercury must be placed with metals that are not migratory.

## Lead

Lead is one of the least soluble of the common metals, although the chloride is fairly soluble and the sulphide is dissolved in sulphuric acid in the presence of an oxidizing agent. The oxidation is attended by the formation of the relatively insoluble lead sulphate, anglesite. This coats the sulphide and tends to delay solution. Deposits with ores carrying galena will generally contain oxidized lead minerals near the surface. It is not uncommon to find galena partly altered to anglesite in the outcrops of deposits containing lead sulphide. Lead minerals, because of their low solubilities, are useful indicators of mineral deposits, and at many places have led to the discovery of silver, gold, and zinc deposits. Like deposits of other difficultly soluble metals, lead deposits are likely to be enriched by removal of valueless materials rather than by concentration in depth by the processes of solution and precipitation.

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[15] T. M. Broderick: Some Experiments Bearing on the Secondary Enrichment of Mercury Deposits, *Economic Geology* (1916), **11**, 645–651.

### Antimony

Antimony dissolves very slowly in the oxidizing zone. Its sulphide deposits form the oxides which are about as stable as the secondary lead minerals, cerusite and anglesite. In depth, however, antimony minerals dissolve readily in alkaline solution. The antimony salts unite with silver-bearing solutions and precipitate antimony sulphosalts of silver. Antimony is not to be classed as a readily migratory metal though it plays an important part in the precipitation of the secondary silver minerals.

### Arsenic

Arsenic dissolves rather readily in acid solutions, but if it is present as a salt of $H_3AsO_3$ much water will hydrolize it, and if present as salts of a base-forming element these also are hydrolized. In an acid solution its salts are oxidized but do not migrate extensively. In alkaline solutions, however, they are easily dissolved and in depth the activities of arsenic in connection with the secondary enrichment of silver are important.

### Bismuth

Bismuth is more or less closely allied to antimony and arsenic. Like them it forms oxides near the surface in its deposits. There is no evidence of its extensive migration. Reasoning from its chemical relations it is not likely that important secondary zones formed by the migration of bismuth will be found. Its sulphate hydrolizes readily in water and in weak acid, forming oxides, and its other salts also hydrolize so readily that they would not be transported.

### Tin

Tin is one of the inert metals. The stannous salts, both sulphate and chloride, are easily soluble, but the stannic salts are readily hydrolized. Thus, in an oxidizing environment such as obtains in sulphide ores near the surface of the earth, the solution of tin is exceedingly difficult. The stannous salts when formed are oxidized to stannic salts and stannic salts will break down almost at once to the insoluble oxide, cassiterite.

Thus tin, although it may go in solution, will be almost immediately precipitated. Experiments by J. P. Goldsberry in the geological laboratory of the University of Minnesota have shown that cassiterite and stannite are both practically insoluble in tenth normal sulphuric acid and hydrochloric acid. Doelter observes that tin oxide is slightly soluble in water. This statement was found by Goldsberry to be erroneous. The method of determination would have detected one part in a million. At

the end of one month, tenth normal sulphuric acid, in contact with stannite and cassiterite, showed only faint traces of tin dissolved. Tin is enriched but little by migration in its deposits, because of the hydrolysis of its stannic salts which, as above stated, are precipitated as the insoluble oxide.

### Chromium, Molybdenum, Tungsten and Uranium

The metals—chromium, molybdenum, tungsten, and uranium—are closely affiliated chemically and all form acid trioxides like that of sulphur, $SO_3$. Chromium forms also the very stable basic oxide $Cr_2O_3$ which resembles ferric oxide in its properties. It forms in silicate environment and is a constituent of igneous rocks. In sulphide deposits chromium is dissolved and reprecipitated, but so far as any evidence is available it is not very migratory. Molybdenum and tungsten oxidize slowly to secondary minerals which generally remain near the parent primary minerals.

Molybdenite is not dissolved in hydrochloric acid nor in sulphuric acid, nor, at the end of one month, in the presence of ferric salts.

Tungsten forms tungstic acid, $H_2WO_4.H_2O$. This is somewhat soluble in water. In moist climates, it could be leached out of deposits which were exposed long enough, but the reaction is exceedingly slow. Tungsten minerals are so insoluble that they commonly form placers. Both molybdenum and tungsten are classed with the non-migratory metals.

Uranium, on the other hand, is carried in underground waters and doubtless forms secondary deposits of considerable importance. The deposits of southwestern Colorado, as already stated, are believed to be reconcentrations by ground water near the present surface.

# The Pyritic Deposits Near Röros, Norway

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(St. Louis Meeting, October, 1917)

## Introduction

Bodies of pyritic ore in schistose rocks have long been known in different parts of the world. The several occurrences resemble each other in being usually of more or less lenticular shape, inclosed in walls of schist or gneiss, and carrying pyrite, chalcopyrite, and pyrrhotite in varying proportions, as the chief ore minerals. The deposits differ, however, in that sometimes one, sometimes another, of the three sulphides mentioned may predominate. The orebodies in general are more or less closely conformable with the schistosity of the wall rocks, although in cases this may not hold true. The boundaries may or may not be sharp, but in general are fairly distinct. The lenses vary in size, may occur singly, or in groups; in the latter case they may be in line, overlapping, or occasionally parallel. Pinching and swelling of individual lenses is not uncommon.

The genesis of these pyritic bodies has provoked much discussion, and they have been variously classed as sedimentary, igneous intrusions, hydrothermal replacements or impregnations, etc. Indeed, in some cases different geologists have most positively assigned widely diverse origins to the same deposit, as in the case of Rio Tinto[1] and Rammelsberg.[2] It is true that in the light of modern criteria, we have been able in a few cases to decipher clearly the process of origin and so there seems no doubt now regarding the genesis of the deposits at Rio Tinto, Spain;[3] Rammelsberg, Germany;[4] Ducktown, Tenn.;[5] southwestern Virginia;[6]

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[1] A. M. Finlayson: *Economic Geology* (1910), **5**, 357.

[2] J. H. L. Vogt: *Zeitschrift für praktische Geologie* (1894), 173; W. Lindgren and J D. Irving: *Economic Geology* (1911), **6**, 303.

[3] Finlayson: *Loc. cit.*

[4] Lindgren and Irving: *Loc. cit.*

[5] W. H. Emmons and F. B. Laney: U. S. *Geological Survey, Bulletin* 470 (1911), 151.

[6] W. H. Weed and T. L. Watson: *Economic Geology* (1906), **1**, 309.

eastern Quebec;[7] and Kyshtim, Russia.[8] The third and fourth are predominantly pyrrhotite, the others mostly pyrite. There are others, however, regarding which we may be supposed to lack positive proof, such as Bodenmais, Bavaria;[9] and Saint Bel, France;[10] while even the great series of Norwegian occurrences are still a subject of dispute.

The present paper deals with certain of these occurrences which are being worked near Röros, Norway, and from which a suite of specimens was collected by the senior author during a visit to that locality in the summer of 1914, the results derived from the study of which it is hoped may throw some further light on the problem of their origin.

The Norwegian pyritic deposits are widely distributed.[11] In the northern part of Norway they are known around Birtavarre, Sulitjelma, Bossmo, etc.; in central Norway around Ytterö, Meraker, Meldalen, Killingdal, Kjöli, Röros, Röstvangen, Foldal, etc.; in southern Norway in Söndhordland and Vigsnäs. The deposits are said to occur at various horizons of the Cambro-Silurian, and in every case in areas of regional metamorphism. These metamorphic rocks are part of an uplifted area, which extends from the British Isles to northern Norway, and was raised at the end of the Silurian to form the Norwegian mountains. The metamorphosed Paleozoics include gray to green phyllites, conglomerates, sandstones, clay slates and limestones, but the wall rock of the ores in most cases is phyllite. The intrusive rocks found within the area range from soda granites to gabbros or peridotites (Vogt), and some of the igneous rocks, such as the granulites, seem to have been intruded during the period of folding.[12] The igneous rocks may be altered, the gabbro especially being changed to the saussurite type.

The orebodies are practically all flat lenses, of varying dip, occurring usually between schist walls, but sometimes within shear zones in the gabbro. While the lenses conform in a general way to the schistosity of the metamorphic rock, they may sometimes cut across it, or even send out stringers across the foliation. Moreover, in some occurrences a breccia of ore and wall-rock fragments has been noticed. A noteworthy feature is that in nearly every case of ore in phyllite, a gabbro mass is not far away.

## The Röros Deposits

The mines visited were the Sextus (Fig. 2) and Kongens (Fig. 1), situated about 8 miles (12.88 km.) west of north from Röros. In addi-

[7] J. A. Bancroft: *Dept. Col'n. Mines and Fisheries, Mines Branch, Quebec,* 1915.
[8] A. W. Stickney: *Economic Geology* (1915), **10**, 593.
[9] E. Weinschenk: *Zeitschrift für praktische Geologie* (1900), 65.
[10] deLaunay: *Ibid.* (1901), 161.
[11] Vogt, Krusch and Beyschlag: *Ore Deposits* (Translation), **1**, 304.
[12] O. Falkenberg: *Zeitschrift für praktische Geologie* (1914).

tion, specimens were also obtained from the Storvarts mine, although this was not visited. The rocks and ores described come mainly from the two first-named mines.

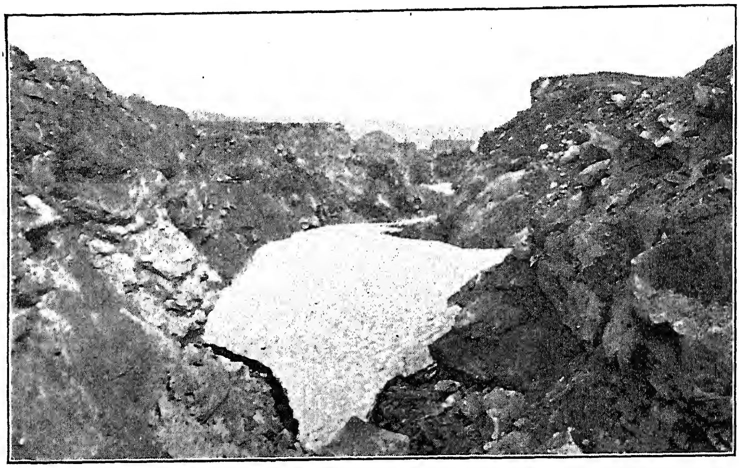

FIG. 1.—VIEW ALONG OUTCROP OF KONGENS OREBODY, SHOWING CUT FROM WHICH ORE WAS REMOVED, GABBRO WALL ON RIGHT AND PHYLLITE WALL ON LEFT.

*Associated Rocks*

*Schist.*—The schist associated with the orebodies at the Sextus and Kongens mines is of a greenish-gray color and fine texture. While usually thinly foliated because of a predominance of chlorite, it becomes in

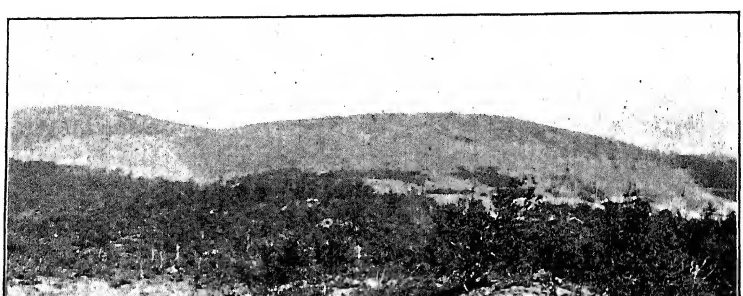

FIG. 2.—VIEW LOOKING TOWARD HILL IN WHICH SEXTUS OREBODY LIES.

places more massive, owing to greater abundance of quartz. One specimen showed small needle-shaped crystals of actinolite. Some of the schist shows strong crumpling, and also lenticular quartz veins, which partake of the folding.

Under the microscope (Fig. 3), the schist is seen to consist usually of an intergrowth of fine-grained quartz, chlorite, brown and green biotite, and

FIG. 3.—THIN SECTION OF CHLORITE, BIOTITE, QUARTZ SCHIST. ORDINARY LIGHT. × 40.

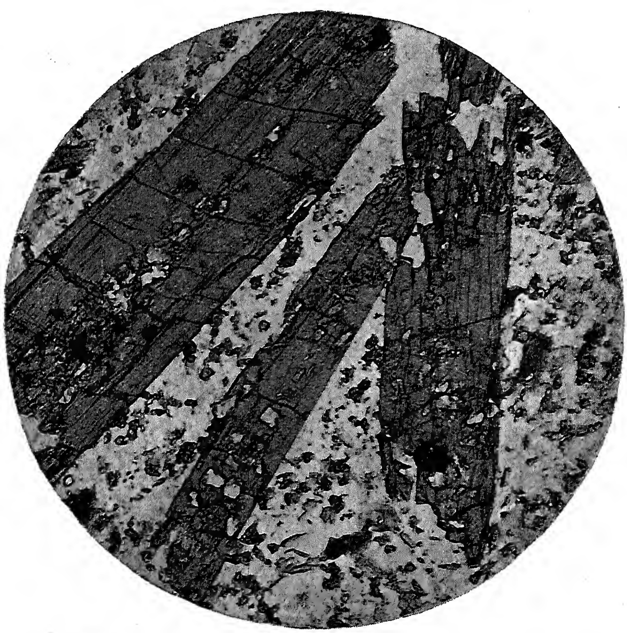

FIG. 4.—THIN SECTION OF SAUSSURITIZED GABBRO, SHOWING POIKILITIC INTERGROWTH OF LARGE LATH-SHAPED AMPHIBOLES WITH QUARTZ, IN FINER-GRAINED GROUNDMASS. ORDINARY LIGHT. × 40.

sometimes actinolite, the bladed minerals showing a parallel arrangement. Epidote and plagioclase are not uncommon, while zoisite, apatite, rutile, magnetite and pyrite were noted. There seems no doubt that chlorite is a normal constituent of the rock and not introduced by hydrothermal action preceding ore deposition, because of its manner of intergrowth with the other common silicates of the schist.

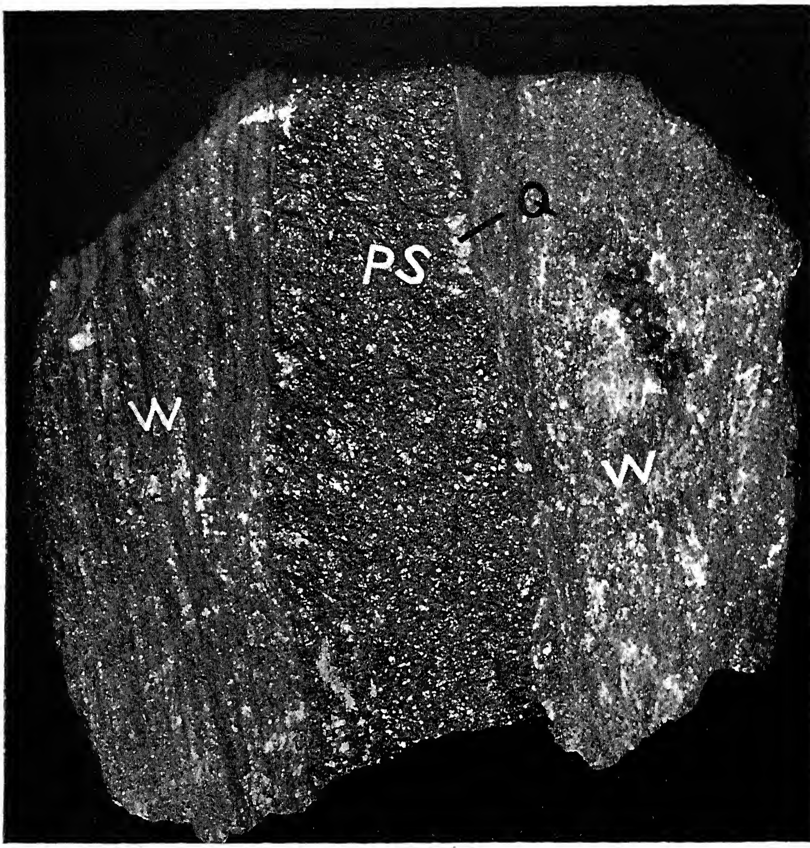


FIG. 5.—HAND SPECIMEN OF QUARTZOSE SCHIST, WITH BAND OF PYRITE AND SPHALERITE (PS), SHARPLY SEPARATED FROM WALL ROCK (W). NOTE SMALL LENS OF LATER QUARTZ (Q). × ¾.

*Saussurite-gabbro.*—No gabbro is found in contact with the ore at the Sextus mine, but at the Kongens a mass of the intrusive lies not far from the sulphides, and, as mentioned later, is in actual contact with them at one place.

Macroscopically the saussurite-gabbro is a greenish-gray, dense, fine-grained rock, showing hornblende needles of varying size, the largest being ¾ in. long and ⅛ in. wide, while the smallest were scarcely noticeable. Under the microscope (Fig. 4), the gabbro exhibits prominent


FIG. 6.—POLISHED ORE SPECIMEN, WITH PREDOMINANT PYRITE SHOWING CRYSTAL BOUNDARIES, AND SPACES FILLED IN BY SPHALERITE (S) AND CHALCOPYRITE (CP). × 35. A COMMON TYPE OF ORE.


FIG. 7.—POLISHED ORE SPECIMEN SHOWING RESORBED PYRITE (P), SURROUNDED BY CHALCOPYRITE (CP) AND SPHALERITE (S). × 35. A COMMON ORE TYPE.

lath-shaped crystals of a blue-green amphibole, imbedded in a fine groundmass composed of quartz with lesser amounts of chlorite, epidote, biotite and magnetite. Small rounded grains of quartz occur in the amphibole as poikilitic intergrowths.

### *Ore*

Vogt in an earlier paper on the Norwegian pyritic[13] deposits, recognized three types of ore, viz.: (1) Chiefly pyrite, usually with some chalcopyrite and blende, but generally with pyrrhotite and only a trace of galena; (2) pyrite and chalcopyrite, with comparatively much quartz and horn-

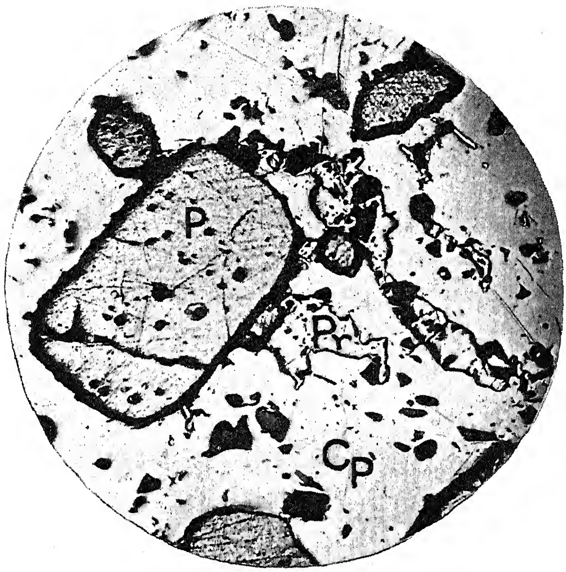


FIG. 8.—POLISHED ORE SPECIMEN SHOWING RESORBED PYRITE (P), IN GROUNDMASS OF CHALCOPYRITE (CP) WITH SOME PYRRHOTITE (PR). × 35.

blende; and (3) pyrite with chalcopyrite, quartz, etc. Falkenberg,[14] on the other hand, says that in the Röros district, particularly, the ores are: (1) Copper-bearing pyrite and (2) copper-bearing pyrrhotite.

Such a grouping, it seems to the authors, is likely to create the impression that these ore types are more sharply defined than they really are. In the Sextus and Kongens mines, pyrite, chalcopyrite or even pyrrhotite may be the predominating sulphide in one or another part of the orebody, but the several types appear to grade into each other, and do not seem to be confined to any particular mine or locality.

At both the Kongens and Sextus mines, the ore forms low dipping

[13] *Zeitschrift für praktische Geologie* (1894), 41.
[14] *Loc. cit.*

lenses enclosed in schist, and at the latter mine, there was a second or upper lens which had been worked out. The gabbro lies above the ore, and is not in contact with it, except in the Kongens mine, where, at one place on the outcrop, one wall of the cut is schist and the other is gabbro (Fig. 1). The ore here is said to have been in actual contact with the intrusive, but as the latter showed evidence of strong folding, and the surface of it was slickensided, the contact may have been a structural rather than an intrusive one. The ore lenses which are usually conformable with the phyllite, pinch and swell, and at times inclusions of the country rock are found in the ore. Quartz lenses are also scattered through it, but are probably of later origin.

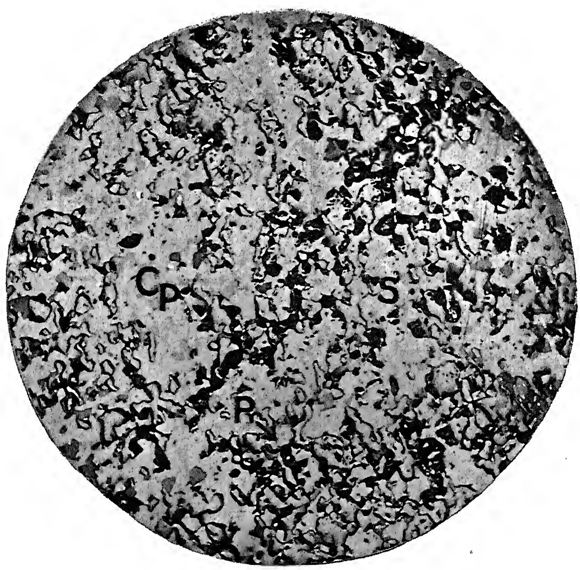


FIG. 9.—POLISHED ORE SPECIMEN SHOWING FINE-GRAINED INTERGROWTH OF CHALCOPYRITE (CP.), SPHALERITE (S), AND PYRRHOTITE (PR). × 35.

The ore boundaries are fairly sharp, and bands of sulphides may occur in the wall rock. These are conformable to the schistosity, and also show a fairly sharp boundary (Fig. 5).

Some specimens from the Sextus mine show an irregular mixture of sulphides and crumpled schist, the former being chalcopyrite, which appears to have been injected prior to the crumpling, as the sulphide shows slickensiding. At one place in the same mine, a fracture about 5/8 in. in thickness was observed to cut the orebody from roof to floor. It was lined with small pyrite cubes, on top of which were larger pyritohedrons, which were in turn coated with calcite. It seems not unlikely that this fissure represents a shrinkage crack, on whose walls the small pyrite cubes crystallized at the same time as the sulphides in the orebody. The

pyritohedrons were probably deposited later by circulating waters. The texture of the ore ranges from fine-grained, granular, massive pyrite, to coarse-grained ore which consists of pyrite crystals, embedded in a groundmass of what macroscopically seems to be chiefly massive chalcopyrite.

In order to study more closely the minerals in the ore, as well as their relation to the wall rock, a number of thin sections and polished specimens were prepared.

Polished surfaces of the massive ore, examined by reflected light, show pyrite, chalcopyrite, pyrrhotite and sphalerite as the metallic min-

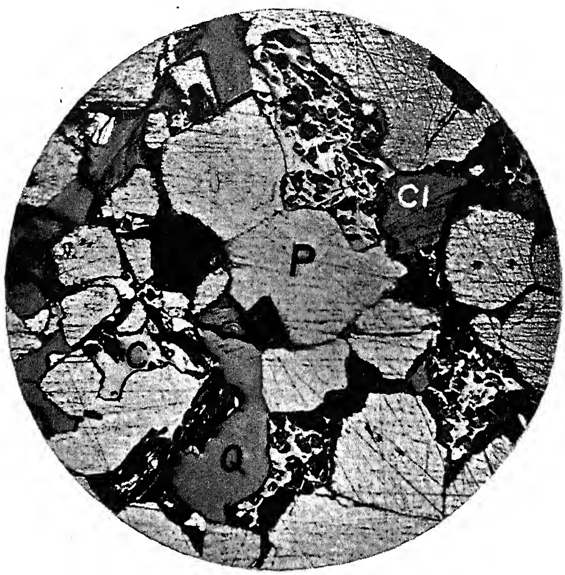


FIG. 10.—POLISHED ORE SPECIMEN SHOWING PYRITE (P), CORRODED BY CHALCOPYRITE (CP) AND SPHALERITE (S); (CL), CHLORITE, (Q), QUARTZ. × 35.

erals.[15] The pyrite is undoubtedly the oldest. It occurs in the form of individual grains, which are bounded by crystal faces where the pyrite is abundant (Fig. 6), and yet had room to grow, but show a rounded outline (Figs. 7 and 8) due to resorption when the other sulphides are greatly in excess of their eutectic with pyrite. The three other sulphides are younger than the pyrite, but in most cases do not show any well-determined order of crystallization among themselves, the boundaries being more or less mutual.[16] The sphalerite and chalcopyrite probably began to separate out before all the pyrite had crystallized, because it

[15] Magnetite, arsenopyrite and galena reported by other observers from the Norwegian deposits were not found in any of the specimens examined by us.

[16] Falkenberg gives the order of formation as pyrite, chalcopyrite, pyrrhotite, sphalerite, magnetite, bornite, chalcocite, galena. Stutzer, however, believes that arsenopyrite and magnetite precede the pyrite.


FIG. 11.—POLISHED ORE SPECIMEN SHOWING FRACTURED PYRITE (P), WITH THE CRACKS FILLED WITH SPHALERITE, CHALCOPYRITE, AND PYRRHOTITE. SURROUNDING MATERIAL MOSTLY PYRRHOTITE WITH SOME CHALCOPYRITE. × 35.

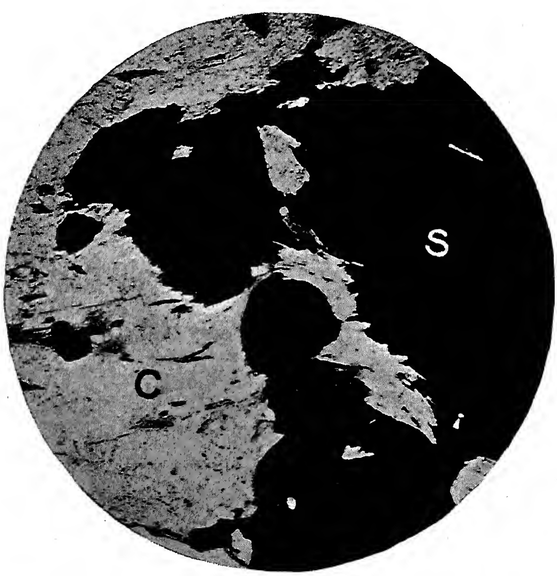


FIG. 12.—THIN SECTION OF LEAN ORE SHOWING SULPHIDES (S), REPLACING CHLORITE (C) AND PENETRATING ALONG ITS CLEAVAGE PLANES. × 40. THIS IS CUT FROM A SPECIMEN OF CRUMPLED SCHIST FROM MARGIN OF SEXTUS OREBODY.

contains inclusions of them, but the pyrrhotite does not appear to have begun to crystallize so early, and in one specimen inclusions of sphalerite and chalcopyrite were found in it. Inclusions of pyrrhotite in pyrite are extremely rare. At times the three later sulphides show a rather fine-grained even intergrowth (Fig. 9), but otherwise the chalcopyrite always forms larger areas. Of these three the chalcopyrite is the most abundant, with sphalerite next and pyrrhotite last. Occasionally, however, the pyrrhotite may form most of the ore.

The later formation of the chalcopyrite, sphalerite and pyrrhotite is


FIG. 13.—POLISHED ORE SPECIMEN SHOWING INCLUDED FRAGMENT OF SCHIST (G), PENETRATED BY CHALCOPYRITE (CP), SPHALERITE (S) AND PYRITE (P). SURROUNDING GROUNDMASS IS OF CHALCOPYRITE WITH SPHALERITE AND PYRRHOTITE. × 35.

clearly shown in two ways, *i.e.:* (1) by the corrosion of the pyrite by the other sulphides (Fig. 10) and (2) by the entrance of these sulphides along fractures in the pyrite (Fig. 11). Where pyrite is very abundant, little or no corrosion has occurred, but where the other sulphides predominate this feature is pronounced (Fig. 8). It consists in the formation of embayments in some, and the rounding of corners of the grains in others.

A fracturing of the pyrite has in some cases occurred before the other sulphides crystallized, because they fill the cracks, but the fissuring seems to have continued subsequent to the solidification of all the sulphides.

Included fragments of the schist, as well as the schist walls of the ore-body, are replaced by the sulphides (Fig. 12), and also penetrated along

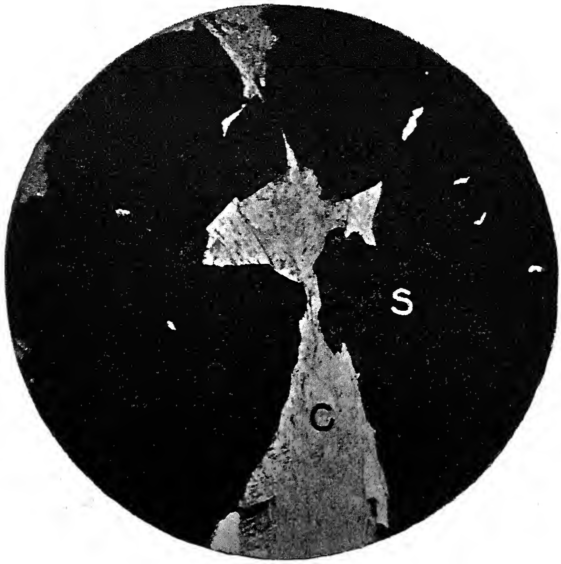


FIG. 14.—THIN SECTION OF LEAN ORE FROM EDGE OF SEXTUS OREBODY SHOWING SULPHIDES (S) REPLACING CHLORITE (C). × 40.


FIG. 15.—THIN SECTION OF WALL ROCK NEAR ORE IN STORVARTS MINE, SHOWING DOLOMITE (D) REPLACING CHLORITE (C) AND BIOTITE (B); ALSO SULPHIDES (BLACK) REPLACING DOLOMITE AND CHLORITE. × 40.

cleavage planes by tongues of the metallic minerals. Of the several sulphides present, the chalcopyrite shows a much greater tendency to send out stringers along the cleavage planes of the chlorite (Fig. 12), than the others. The schist fragments within the ore are not oriented, and, moreover, they are often bent, twisted and shredded (Fig. 13).

Quartz grains are found in the ore. They at times seem to be contemporaneous with the sulphides, but in most cases they are genetically related to the chlorite and other minerals of the schist and seem to be of similar age. The former relationship of quartz to sulphides was also noticed by Falkenberg in the deposits at Lokken.[17] But small lenses or eyes of quartz were also found (Fig. 5).


FIG. 16.—THIN SECTION OF LEAN ORE FROM NEAR MARGIN OF KONGENS ORE-BODY, SHOWING SULPHIDES (S) REPLACING QUARTZ (Q) AND CHLORITE (C). × 40.

Some ore specimens show an indistinct banding due to the predominance of one of the later sulphides along certain lines, and corrosion of the pyrite is more noticeable in those bands in which sphalerite is more abundant.

*Relationship of Sulphides to Non-metallics*

It has already been pointed out in describing the schist country rock that the chlorite, biotite, garnet, feldspar and quartz, were contemporaneous. Study of thin sections shows that the schist may contain vein-

[17] *Zeitschrift für praktische Geologie* (1914).

lets of later dolomite (Fig. 15) and quartz, both of which minerals can be seen replacing at least the chlorite (Fig. 15). Later than any of these are the ore minerals, and their relationship to the silicates is extremely interesting, where it can be studied. This, of course, is chiefly along the boundary of the orebody where it comes into contact with the wall rock (Figs. 12, 14–17). To the naked eye, the contact between the two appears quite sharp, but even under low powers the polished rock shows the sulphides extending irregularly and for a short distance into the schist. The relationship between sulphides and schist can also be studied where

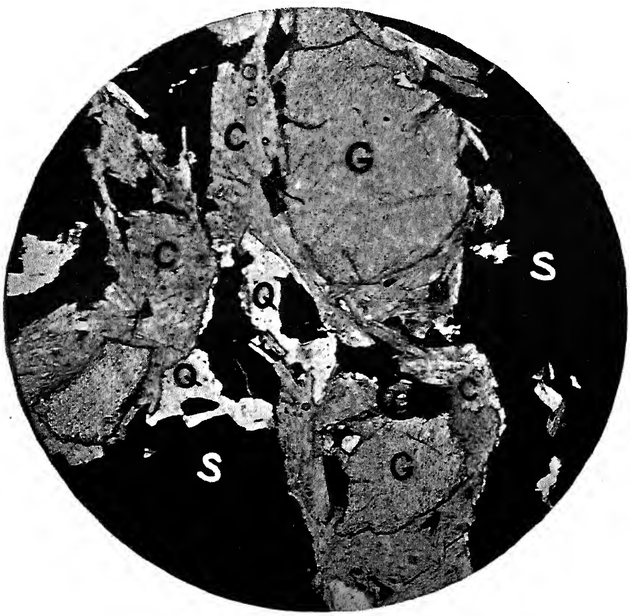


FIG. 17.—THIN SECTION OF WALL ROCK CLOSE TO ORE (SEXTUS MINE) SHOWING SULPHIDES (BLACK) REPLACING GARNET (G), CHLORITE (C) AND QUARTZ (Q). × 40.

the former surrounds and penetrates inclusions of the latter (Fig. 13). In some cases the sulphides form films between the cleavage plates of the chlorite, but at others they can also be distinctly seen replacing minerals of the schist such as chlorite (Fig. 16), biotite, garnet (Fig. 17), dolomite (Fig. 15) and quartz.

Careful search of all the thin sections failed to reveal any sericite, and in this connection it is interesting to note that all the feldspar grains observed were unaltered. Brown biotite is sometimes found in small grains contemporaneously intergrown with the sulphides.

Another important point to be stated is that the presence of chlorite does not seem to be dependent on the processes of ore introduction, since

it is found in both the mineralized and unmineralized schist in its normal relationship to quartz and the other silicates.

## Origin

*Previous Views.*—Many theories have already been expressed regarding the origin of these ores. The earlier writers seemed to lean strongly toward a sedimentary origin, including Helland[18] in the seventies and later in Germany, Stelzner,[19] Klockman,[20] as well as others. They based their theory on the apparently uniform concordance of the orebodies with the foliation of the schist, the fact that orebodies and schist had (they thought) been folded together, and apparent banding in the ore. The lens-shaped nature of the orebodies was explained by some as due to their being beach deposits. Vogt also at one time regarded the ores as chemical precipitates derived from submarine exhalations emanating from the gabbro. Reusch suggested the possibility of the sulphides having replaced limestone, but there is no field evidence to bear this out. Sjögren,[21] after studying Sulitelma, advanced the somewhat curious view that iron and copper solutions derived from the weathering of pyrite in the gabbro, and containing also alkali carbonates from the same source, had percolated downward and come into contact with rising waters carrying hydrogen sulphide, resulting in a precipitation of the iron and copper sulphides. He even suggested that this process might apply to other Norwegian occurrences.

At the present day, most geologists no doubt believe that the Norwegian pyrite deposits like those of the Roros and Sulitelma type are of epigenetic character. This theory was early expressed by Kjerulf,[22] and the earlier French investigators[23] Durocher and Duchanoy, who believed that we must regard the pyrite as intruded, and that the ore followed the planes of schistosity because it was the path of least resistance.

Stutzer,[24] after studying Sulitelma, came to regard the deposits as *injection veins* (*injections-gänge*), and in this connection pointed out that there may be every gradation between a watery solution containing dissolved minerals from a magma, and a magma containing a comparatively small amount of water. He furthermore drew attention to the

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[18] Vorkommen von Kiese in gewissen Schiefern in Norwegen, *Universitàts Programm,* 1873.

[19] *Die Sulitelma Gruben in nordlichen Norwegen,* Freiberg, 1891.

[20] *Zeitschrift der Deutschen Geologischen Gesellschaft* (1893), **45**, 281.

[21] *Geol. Foren. Forh,* November, 1893.

[22] *Geologie des sudlichen und mittleren Norwegen,* 1880.

[23] *Annales des Mines,* ser. 4 (1849), **15**, 171, 267 and ser. 5 (1854), **5**, 181.

[24] *Zeitschrift fur praktische Geologie* (1909), **17**, 130, and *Oesterreichische Zeitschrift ür Berg. und Hutten Wesen* (1906), No. 44, 567.

fact that during the crystallization of a magma the metallic minerals may separate and segregate at practically the same time as the silicates, or they may segregate and crystallize out a little later, but before doing the latter, wander into the already crystallized part of the magma or even the wall rock, giving what Sjögren named *diamagmatic ores*. He therefore regards the sulphide bodies like those of Sulitelma and Röros as of this class.

Falkenberg,[25] after a somewhat extensive study of the deposits of southern Norway, evidently considers many of them to be of eruptive origin. For he says that the common forking of many of the sulphide orebodies, which is closely associated with brecciation, can be explained only on eruptive grounds. The corrosion phenomena (also noted by us) he believes explainable only by crystallization from fusion. After commenting on the close association of the orebodies and gabbro, he remarks that he considers it difficult to state definitely the nature of the magma, but believes that no doubt water and a small amount of silica reduced the viscosity of the molten or water solutions of high temperature, thus permitting them to ascend the fissures of the surrounding rock.

Lindgren[26] expresses the view that some deposits of solid pyritic minerals present features which can hardly be explained otherwise than by actual injection of molten sulphides, perhaps to be considered as residual solutions from adjoining intrusive bodies.

Vogt, Krusch and Beyschlag,[27] after emphasizing the close relationship of most of the sulphide orebodies with the gabbro, make the statement that the formation of the pyrite bodies is contemporaneous with the magmatic period of the eruptive rocks, and that it follows from the form and structure of the deposits that they could not have been formed by long-continued deposition from different solutions, but that the whole of the material must have been introduced or injected at once. Furthermore, that these pyrite deposits are products of magmatic differentiation. They agree with Brögger in believing that the rounded idiomorphic pyrite crystals not only show resorption, but conclude that this was of magmatic character.

*Authors' Theory.*—The theory of a sedimentary origin hardly seems any longer tenable, and we must therefore recognize that these orebodies are of later age than the enclosing rocks. It still remains, however, to be explained whether they are veins or dikes, a problem by no means easy, because of the lack of well-defined and never-failing criteria for identifying these two processes.

If the orebodies are veins, one might expect to find some good evidence of hydrothermal alteration of the walls, such as the development of

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[25] *Zeitschrift fur praktische Geologie* (1914), **22.**

[26] *Mineral Deposits,* 768.

[27] *Ore Deposits,* **1,** 312.

chlorite or sericite. It is true that chlorite is found in the rock associated with the sulphides, and even replaced by them, but its relationships there are so identical with those in the unmineralized schist, that we can see no good reason for regarding it as secondary, in the sense that it is a product of vein-wall alteration, and not of regional metamorphism. The absence of sericite is also significant, for there is feldspar present which could have yielded the necessary elements for its development. These facts, therefore, would seem to eliminate one of the most strongly held criteria for vein formation by deposition from solution.

In view of the fact that the ore contains at least two minerals (pyrrhotite and biotite) which are characteristic of high-temperature conditions, it hardly seems likely that these materials could have been introduced in hot aqueous solutions without some hydrothermal alteration of the walls.

The alternative view would therefore be to regard these orebodies as dikes. This idea, as applied to the formation of sulphides, has perhaps been regarded as a somewhat radical one, but in our opinion is not at such great variance with the commonly accepted view of vein formation by precipitation from solution. Any magma is a natural mixed solution of rather complex character. It may differentiate, and the lighter constituents tend to rise, while the heavier sink either in a subcrustal reservoir or in an intrusive body such as a sill. These lighter constituents include the water and other vein-forming compounds, which escape from the magma during cooling, and may form veins in the surrounding rocks. Portions of the remaining magma may make their way into fissures and form dikes of either acid or basic character, and if very basic they might be composed almost entirely of sulphides or metallic oxides.[28] In any case, however, it is a deposit from solution, and essentially the difference lies in the materials present.

We have come to say that a solution with much water in it deposits a vein. It comes from the top of the magmatic unit,[29] and contains other mineralizers also, which alter the country rock. On the other hand, there may be a solution that gets a favorable chance to leave the very bottom of the unit, and be composed almost wholly of metallic compounds. It does not alter the country rock, because most of the silica, water, etc., are elsewhere. But it may form a fissure deposit, solidify according to the laws of solutions, and replace walls or inclusions with its own minerals. Even such a basic mass as a mixture of sulphides might contain a little water or other mineralizers, and this may be borne out by the finding of small and scattered amounts of such minerals as tourmaline, apatite, cordierite, etc., by other investigators, although one cannot always be

[28] Iron Mountain, Wyoming; or Kiruna, Sweden.

[29] B. S. Butler: *Economic Geology* (1915), **10**, 101.

sure from their descriptions whether they are ingredients of the metamorphic rocks, developed independently of ore-forming processes.

There seems to us then no serious objection to the theory that the deposits of Röros and similar Norwegian ores have originated as outlined above, and represent injections of magmatic sulphides coming probably from the same reservoir as the gabbro.

*Criteria to be Applied.*—Having expressed our opinion regarding the origin of these orebodies, it may be well to consider in a little more detail some of the criteria by which such deposits are to be judged. It has been stated that so far as the Röros occurrences mentioned are concerned there is no evidence of hydrothermal action on the walls, and this is at


Fig. 18.—Polished ore specimen from Ascot, Quebec. Shows pyrite (P) corroded by sphalerite (S), in groundmass of chalcopyrite (Cp), sphalerite, and quartz (Q). × 35.

least negative evidence in favor of the view expressed. Since the orebodies do not represent magmatic segregation and crystallization in place, we cannot look for such criteria as primary intergrowths with silicates of the original magma. Corrosion of pyrite by other sulphides, though regarded by some as evidence of molten conditions, is not necessarily such, since the same may be observed in pyritic deposits formed by hydrothermal replacement. Fig. 18 shows a case of this from the King mine at Ascot, Que., photographed from a polished specimen kindly supplied us by Prof. J. A. Bancroft of McGill University. Here the pyrite and sphalerite have been deposited by replacement of schist, the sphalerite, however, having corroded the pyrite. A point to be considered is the entrance of molten sulphides into the schist. This has been largely

parallel with the schistosity, and it is possible that the sulphide intrusion had sufficient force behind it to widen the fracture it followed.

So far as the mines visited at Röros are concerned, there were few observed inclusions of wall rock in the ore, although at several other localities they have been noted, and in some cases a breccia of rock fragments in a matrix of ore has been mentioned. In our polished specimens, detached grains of schist were not infrequently observed (Fig. 13), and often in such positions as to indicate that they had been pried off the wall. This fact was also noted by Stutzer, and some of the illustrations given by Vogt produce the same impression. Such scattered and unoriented fragments would be in accordance with the injection theory.

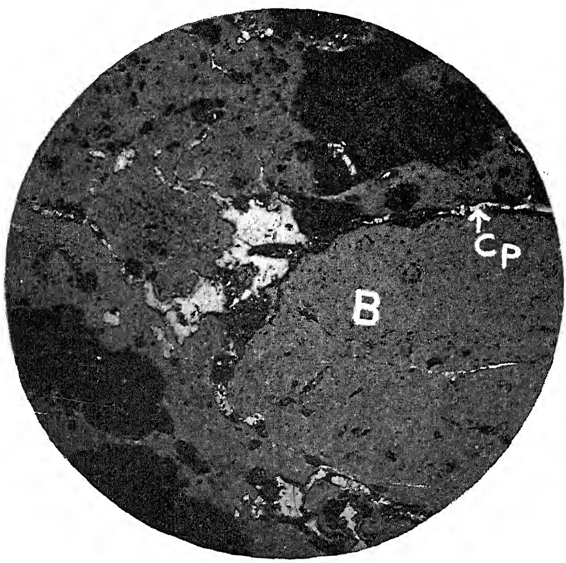


FIG. 19.—POLISHED SURFACE OF FIRE BRICK (B), SHOWING PENETRATION OF MOLTEN SULPHIDE, CHALCOPYRITE (CP) ALONG MINUTE CRACKS. × 35. THE VEINLET INDICATED BY ARROW IS ABOUT 1/50 MM. THICK.

Both these fragments (Fig. 13), and also the schist of the wall itself, showed minute filaments of sulphide, entering between the cleavage flakes, of the chlorite, and it might be thought that molten sulphides could hardly be sufficiently fluid to penetrate the rock in such thin filaments. This, however, is by no means impossible, for Stutzer has described the penetration of gneiss by veinlets of molten sulphide,[30] and in Fig. 19 we give a photomicrograph of a firebrick from the hearth of a furnace at Trail, B. C., which is quite extensively impregnated with chalcopyrite, bornite and chalcocite. The chalcopyrite stringer indicated by the arrow in figure has a diameter of 0.02 mm.[31]

[30] *Zeitschrift für praktische Geologie* (1908), **16,** 119.

[31] See N. B. Davis: *Economic Geology* (1915), **10,** 663.

The lenticular character of the orebodies needs some consideration, and we are not able to state whether it is altogether an original structure. Some features point to its being due to post mineral movement, which caused pinching of the orebodies. That such pressure has probably been exerted is shown by the minute fractures in the pyrite; slickensided surfaces in some of the ore specimens; the folding and smoothing of the gabbro at the outcrop of the Kongens orebody; and lastly, the occurrence of corroded pyrite grains in schist fragments, near the border of the orebody, their relation being such as to suggest that they had possibly been pushed into the schist.

The massive and uniform character of the ore, freedom from bands and


FIG. 20.—POLISHED SURFACE OF ORE FROM ARMINIUS MINE, LOUISA COUNTY, VA., SHOWING PYRITE (P), SURROUNDED BY CHALCOPYRITE (CP) AND SOME QUARTZ (Q). × 35.

grains of wall-rock minerals, while suggesting igneous injection, is not absolute proof of it, for complete replacement may give equally massive ore, and even coarsely granular material, so that there seem to the authors no means of telling from the hand specimens of the material whether or not it was replacement, ordinary vein filling, or igneous injection. In Fig. 20 we give a photomicrograph of a polished specimen of pyrite ore from Louisa County, Virginia, which is a replacement of schist or limestone, and yet so far as its appearance goes it does not differ essentially from some of the Röros material (Figs. 6 and 10).

That the orebodies were formed under conditions of high temperature is indicated by the presence of such minerals as pyrrhotite and biotite,

and there is no doubt of the latter being primary, and not derived from pyrite by metamorphism (as claimed by Klockman), since the latter is not always in contact with the pyrite.

A final point to be mentioned is the mineral composition of the ore, and the objection to calling it magmatic injection because of the presence of pyrite.

Tolman and Rogers[32] in their valuable paper on magmatic sulphides, refer to the Norwegian pyrite deposits, but while they say that they are not personally familiar with them, do not hesitate to reject the view of their being magmatic sulphides. They doubt seriously whether pyrite is a characteristic magmatic mineral, and note that pyrrhotite seems to take the place of pyrite in magmatic ores, which they regard as but natural since pyrrhotite is the iron sulphide stable at high temperature.[33] It is known, however, that pyrite may exist as a primary constituent of igneous rocks, and there seems no doubt that large pyrite bodies have been formed under high temperature conditions, like those of Louisa County, Virginia. These are but a step removed from injection masses, such as the Röros bodies may be, and because pyrite is of minor importance as a magmatic mineral in most orebodies of that type, it is not necessarily to be excluded from all.

## Summary

By way of summary, it would seem that the points favoring the injection theory are:

1. Absence of hydrothermal alteration of walls.
2. Uniformly close association with gabbro.
3. Unoriented inclusions of wall rock in ore.
4. Massive and sometimes porphyritic orebody, not in all cases absolutely conformable with enclosing schist.

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[32] Leland Stanford Junior University Publications, 1916.

[33] E. T. Allen, J. L. Crenshaw and J. Johnston: The Mineral Sulphides of Iron, *American Journal of Science* (1912), **33**, 169.

## The Sulphur Deposits in Culberson County, Texas

WILLIAM B PHILLIPS,* HOUSTON, TEXAS

(St. Louis Meeting, October, 1917)

THE earliest mention of the sulphur deposits in what is now Culberson County, Texas, seems to be contained in "Explorations and Surveys for a Railroad Route from the Mississippi River to the Pacific Ocean, United States War Department, 1853–54." Captain John Pope (he was not commissioned Captain until 1856) had charge of one of the divisions of the Survey. In his report on the geology of the route, William P. Blake noted the analysis of a greenish earthy powder from the bed of Delaware Creek, at the springs, which contained 18.28 per cent. of free sulphur. The report states:

"From the chemical examination of the waters of the spring, by Dr. Booth, it appears that chloride of sodium is the principal solid ingredient, but considerable quantities of sulphide of sodium are present. * * * * * The waters from the springs examined by Dr. Booth were alkaline, but it is possible that some of the springs are acid; and in this case the precipitation of sulphur would ensue on the mixture of the acid waters with those charged with alkaline sulphurets and the origin of the deposits of sulphur would be thus explained."

This is the first reference to the existence of sulphur in that region, the first mention of the character of the water found there and the first suggestion as to the origin of the deposits. The possibility of the existence of acid waters was confirmed by E. M. Skeats, nearly 50 years later.[1] In this publication, Skeats says that the water from the sulphur deposits in Section 11, Block 114, Township 5, contains 79.08 grains of free sulphuric acid per gallon. This locality is about 13 miles (21 km.) south of Delaware Creek and about 35 miles (56 km.) east of the Delaware Springs.

The next published reference to the existence of sulphur in Trans-Pecos Texas is in the First Annual Report of the Geological Survey of Texas, 1889, page 226, where W. H. von Streeruwitz mentions that sulphur is found in that region, but gives no further information. In the

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* Consulting Engineer.

[1] Sulphur, Oil and Quicksilver in Trans-Pecos Texas. *University of Texas Mineral Survey, Bulletin* No. 2 (1902).

Third Annual Report of that Survey, 1891, in his report on the Geography, Topography and Geology of the Llano Estacado, or Staked Plain, page 165, W. F. Cummins says:

"We had been informed that there was a bed of native sulphur in this vicinity (Castile Springs), but after a diligent search for it for two days were unable to find it."

The Castile Spring to which Cummins referred is about 16 miles northwest of Maverick Springs. There are no known deposits of sulphur in this immediate vicinity, but in Sections 8, 16, 17 and 18, Block 61, a few miles west of Castile Spring, there are some very interesting deposits at and near the Grant mines. These have been prospected to some extent but have not been developed.

The next published account of the sulphur deposits in Culberson County is contained in an article by Dr. Eugene A. Smith, State Geologist of Alabama (*Science*, May 1, 1896). This was republished in full in the bulletin of the University of Texas Mineral Survey above referred to. It was condensed from a private report made by Dr. Smith. The appearance of this bulletin, in 1902, marks the first attempt to describe these deposits in some detail.

It contained the article by Dr. Smith, just alluded to, the full report of E. M. Skeats, who had been engaged by the University Mineral Survey to study the deposits and describe them in as much detail as was then possible, and the results of personal observations by the writer, who was at that time Director of the Survey, and whose acquaintance with these deposits has been continued since 1901.

In 1904, by coöperation between the U. S. Geological Survey and the Mineral Survey, George B. Richardson published the results of his observations in that region. His report,[1] which contained a geological map and was a valuable contribution to the literature concerning the district, has been out of print for some time. There are still a few copies of the map, which may be obtained from the Bureau of Economic Geology, University of Texas.

In 1912,[2] 1916,[3] and 1917[4] the writer published articles on the subject of sulphur in Culberson County.

These furnish the principal published data concerning the district, but we understand that a bulletin on this subject, by E. L. Porch, Jr., Bureau of Economic Geology, University of Texas, will soon be available.

[1] Reconnaissance in Trans-Pecos Texas North of the Texas and Pacific Railway. *Bulletin* No. 9, *University of Texas Mineral Survey* (1904).

[2] Sulphur Deposits in Culberson County. *American Fertilizer* (1912), **36,** No. 12, 449 (1912).

[3] The Sulphur Situation in Culberson County, Texas. *Manufacturers' Record*, Baltimore (Dec. 7, 1916).

[4] Development of Sulphur Deposits in Culberson County, Texas. *Manufacturers' Record*, Baltimore (July 19, 1917).

Many private reports have been consulted through the kindness of the persons for whom they were made, and the writer has availed himself of his own copious notes made during a period of 16 years.

## Location of the Sulphur Districts

A few years ago the Legislature of Texas divided El Paso County, forming Culberson County from the eastern portion. This accounts for the early references to sulphur in El Paso County. The principal sulphur districts are from 35 to 40 miles (56 to 64 km.) north of the town of Toyah, Reeves County, and a little further from Pecos, the county seat. These towns are on the Texas & Pacific Railroad west of the Pecos River. The Santa Fe lines also run into Pecos from the north, through Carlsbad and Roswell to Clovis, where connection is made with the main lines of this system.

There are many outlying and separate localities where more or less prospecting has been done, but the principal development has been in the Maverick Springs district, from 15 to 20 miles west of the Santa Fe lines at Orla, Reeves County, 40 miles north of Pecos. The first attempts at mining and extracting the sulphur from any of these deposits were made about 20 years ago by a Mr. Choteau, St. Louis, on what is known as the Cooksey property, in Sections 10 and 15, Block 114. A vertical extractor, for use of steam, was in use there for some time and it is reported that two car loads of sulphur were shipped to St. Louis. Of recent times, however, most of the prospecting and development work has been carried on in the Maverick Springs district, about 15 miles southeast of the Cooksey property. The deposits at and near Maverick Springs have been known for more than 20 years. The old Kyle property (West Texas Sulphur Co.), on which there was a sort of extractor and an attempt to build a calcarone, is within a mile of the Springs, the Dot property is 3 miles to the southwest (the Toyah Valley Sulphur Co. owns a good deal of land near the Dot and also near the old Kyle), while the holdings of the United States Sulphur Co. and the Michigan Sulphur & Oil Co. are toward the head of Virginia Draw, about 5 miles southwest of Maverick Springs.

The properties that up to this time have undergone some prospecting and development in the Maverick Springs district are: The West Texas Sulphur Co., the Toyah Valley Sulphur Co., the Sun Sulphur Co., the United States Sulphur Co. and the Michigan Sulphur & Oil Co. All of these are comprised within an area 5 miles southwest of Maverick Springs.

The American Sulphur Co. is opening a property near Rustler Springs and has produced a small amount of sulphur.

There are other localities in the county where discoveries have been made and where more or less prospecting has been done. The principal ones are: Block 42, Sections 7, 8, and 9, Pat, Lone Wolf and Golson

claims; Block 60, Sections 15 and 16; Block 61, Sections 8, 16, 17 and 18, Grant mines, etc.; Block 109, Sections 4, 5, 9, 10 and 24, Nelson claims, etc.; Block 113, Section 14, the Hicks claims; Block 114, Sections 10 and 15, the Cooksey property; Block 115, Sections 10, 11, 18, 19, 22 and 23. In Reeves County, Block 59, Section 17 or 9, near the old Huling-Ross oil wells and within a short distance of the oil well bored to a depth of 4100 ft. (1249 m.) by The Texas Co.

With the exception of the deposits in the Maverick Springs district and the one last mentioned, none of the discoveries is within 25 miles of a railroad.

## Geology

The Guadalupe Mountains, 45 miles (72 km.) northwest of Maverick Springs, form the chief topographic feature of this entire region. They attain, in El Capitan, an elevation of 8690 ft. (2649 m.), the highest point within the State. Thirty miles southeast of these mountains and separated from them by a rolling plain are the Rustler Hills, which are much lower. From the Rustler Hills east to the Pecos River, the slope is gradual but constant, the elevation at Orla, on the Santa Fe lines, being 2853 ft. (869 m.).

George G. Shumard[4] considered the Guadalupe Mountains as Permian. Jules Marcou[5] and Robert T. Hill[6] classed them as Carboniferous. R. S. Tarr[7] says:

> "No Permian beds appear between the Guadalupe Mountains and the Pecos River in the section studied, but wherever the Carboniferous is covered by later formations these deposits are either Quaternary or Cretaceous."

Walter P. Jenney[8] speaks of the Guadalupe Mountains as Carboniferous. George B. Richardson[9] considers the Delaware Mountains (the extension of the Guadalupe Mountains to the south) as Permian and is inclined to place the sandstones, limestones and black limestones in the Guadalupe as also in the Permian. In this view he is supported by G. H. Girty, whom he quotes.

On the geological map accompanying his report, Richardson marks an area of some 16 miles (25.7 km.) east of the Delaware Mountains as belonging to the Capitan limestone division of the Permian series of the Carboniferous. East of this area again and for some 16 miles he marks

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[4] *Transactions, St. Louis Academy of Science* (1858), 273.

[5] *Geology of North America.*

[6] *Physical Geography of the Texas Region* (1900).

[7] Reconnaissance of the Guadalupe Mountains. *Bulletin* No. 3, *Geological Survey of Texas* (1892), 21.

[8] *American Journal of Science*, 3d Ser. (January, 1874), **7**, 25–28.

[9] Reconnaissance in Trans-Pecos Texas. *Bulletin* No. 9, *University of Texas Mineral Survey* (1904).

the formation as Castile gypsum, questionably belonging to the Permian series of the Carboniferous. This brings one to the Rustler Hills. These he classes with the Castile gypsum, questionably Permian series of the Carboniferous. But Tarr says that no Permian beds occur between the Guadalupe Mountains and the Pecos River, in the section studied, and it would have been difficult for him to have reached the Pecos from the Guadalupe Mountains without crossing the Rustler Hills, which appear to belong among the transition beds between the true Carboniferous and the true Permian, but it is not yet known to which of these formations they are closest kin.

East of the Rustler Hills and coming into and across the sulphur deposits in the Maverick Springs district are the deposits that Richardson

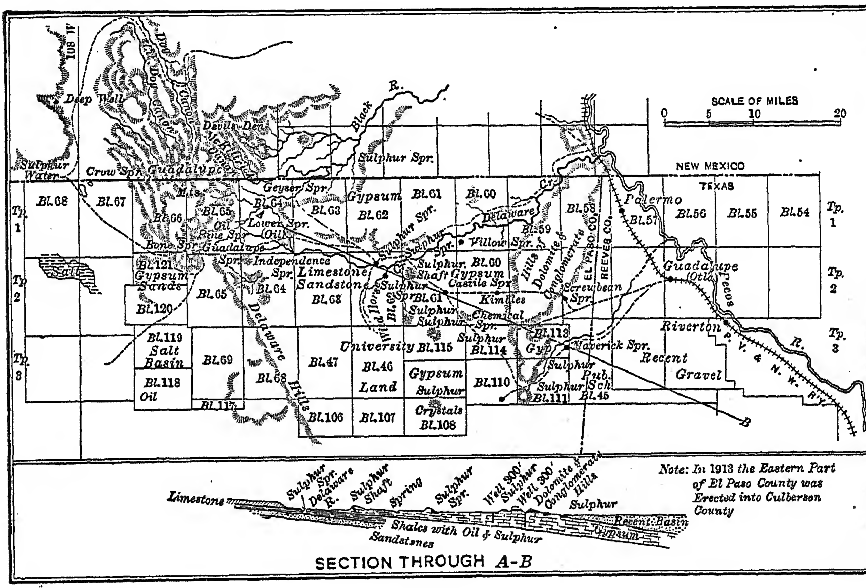


FIG. 1.

classes as Quaternary, Recent and Pleistocene. They are characterized by heavy deposits of gypsum, a few outcrops of "Red Beds" and the remains of dolomite hills. These "Red Beds" are more extensive than Richardson indicates. Instead of occurring over a few acres they are to be observed, at intervals, over a considerable area, especially in Sections 13, 14 and 24, Block 113, and in Section 1, Block 111.

An interesting occurrence in connection with these "Red Beds" was revealed by core drilling a little northeast of the center of Section 1, Block 111, conducted by the writer in November, 1916. This drilling was started in an outcrop of the "Red Beds" which could be traced from

the Virginia Draw eastward to the top of the divide between this Draw and the Maverick Draw. To a depth of 47.5 ft. (14.48 m.) the boring was in red sandstone, which was characterized by many whitish-gray splotches. At 47.5 ft., a bed of massive white gypsum was encountered and in this gypsum were scattered crystals of sulphur. This gypsum was 2.5 ft. (0.76 m.) thick, with red sandstone underneath, to a depth of 74 ft. (22.5 m.) when the boring was discontinued. There was no water in this hole.

On both sides of Virginia Draw, and even in the Draw itself, are outcrops of a dolomite closely resembling the dolomite that Richardson mentions as occurring in Horseshoe Draw, 2 miles to the northwest. Some of the borings in the upper part of Virginia Draw stopped in dolomite after passing through notable beds of sulphur, and sulphur has been found in the dolomite itself and underneath it. Considering the occurrence of this dolomite in immediate association with the sulphur deposits, I am disposed to regard them, in the Maverick Springs district, at least, as of the same age as the Rustler Hills, *i.e.*, Permo-Carboniferous. The sulphur does occur with rounded gravel, etc., of the Quaternary, but this gravel is evidently not in place, in many localities, and the sulphur associated with it may be due to deposition from gases and alkaline sulphides.

As a further contribution to our knowledge of the geology of the immediate region there is given here the log of a well bored to the depth of 2265 ft. (690 m.), near the south line of Survey 24, Block 110. Two wells were bored here by The Texas Petroleum Co., in 1914 and 1915, but one of them was carried down only to a depth of 541 ft. (164.9 m.). These are known as the Rustler Springs wells and I am indebted to Mr. C. R. Troxel, Toyah, for permission to use the log here given. The locality is about 4 miles (6 km.) southwest of the sulphur property of the Michigan Sulphur & Oil Co. and the holdings of the United States Sulphur Co., toward the head of Virginia Draw. It is between the sulphur deposits that are south and west of Rustler Springs and those that are northeast of these Springs, being about equidistant from the two, viz., about 4 miles. The wells are in the Rustler Hills at an elevation of 250 to 300 ft. above the deposits in Virginia Draw. They go far below the sulphur horizon, as now known, the bottom of the deeper well being about 2000 ft. (609 m.) below the level of Virginia Draw. No deposits of sulphur were found in either well, but several barrels of oil were reported from a depth of 2018 to 2020 ft.

Description of samples from Troxel No. 2 well, Rustler Springs, Culberson County, Texas. Located near the south line of Survey 24, Block 110, Public School Lands, close to Troxel No. 1. Drilled in 1915. Samples examined by Dr. J. A. Udden, Bureau of Economic Geology, University of Texas.

Red Sandy clay 85 ft.

| From | To | Feet | Formation |
|---|---|---|---|
| 85 | 150 | 65 | Greenish-gray and some red clay. Some fragments of gray, compact dolomite. |
| 150 | 235 | 185 | Dark gray and brownish-gray limestone of very compact texture. Gypsum, quartz grains, fragments of chert and other quartz and pyrite noted. Little gas. |
| 235 | 240 | 5 | Gray dolomite of very fine texture. Pyrite, gypsum, rounded clear quartz grains and quartz crystals noted. |
| 240 | 250 | 10 | Dark gray dolomite of fine texture. Effervescence in acid is extremely slow. With this is some marl. Fragments of gypsum, many. There are some rounded grains of quartz and some quartz crystals. Much pyrite. No bituminous fumes noted. |
| 250 | 255 | 5 | White gypsum of moderately coarse crystalline texture. The sample contains some double pyramidal crystals of quartz, the largest 0.75 mm. in length, with dark transparency. |
| 255 | 350 | 95 | Mostly gypsum, with some anhydrite, white and gray, mostly moderately coarsely crystalline. |
| 350 | 405 | 55 | Mostly gray and brownish dark gray dolomite of fine compact texture. In thin section it is seen to contain some anhydrite and is impregnated with pyrite, which occurs in exceedingly fine particles. The sample contains coarse rounded quartz sand and fragments of quartz pebbles. Much anhydrite present. |
| 405 | 450 | 45 | Compact gray dolomite and white anhydrite. Bright yellow pyrite noted. |
| 450 | 500 | 50 | Anhydrite and gray compact dolomite. Bright yellow pyrite and fragments of quartz noted. |
| 500 | 515 | 15 | White and gray anhydrite. |
| 515 | 540 | 25 | White anhydrite with some rounded quartz sand grains, pyrite noted. |
| 540 | 545 | 5 | White anhydrite. |
| 550 | 555 | 5 | White and gray anhydrite, some quartz sand, fragments of gray flint, crystals of gypsum, and pyrite. |
| 555 | 560 | 5 | Anhydrite with some sand, some pyrite and considerable chert which seems to have been in the form of pebbles. Most of the chert is dark. Oil colors on slush. |
| 560 | 600 | 40 | White anhydrite, some flint present. |
| 600 | 625 | 25 | White and gray anhydrite. Some quartz, some pyrite and some flint present. |
| 625 | 630 | 5 | White anhydrite, with a few particles of calcareous material (description by E. L. Porch, Jr.). Gray dolomitic limestone containing pyrite, anhydrite and sand. |
| 630 | 650 | 20 | Light gray dolomitic limestone containing anhydrite, pyrite and sand. Other sample consists of anhydrite. |
| 650 | 680 | 30 | Light gray anhydrite containing sand, pyrite and black minerals. Gave off a faint odor of bitumen when heated. An endothyra was noted. Oil colors on slush. |

| From | To | Feet | Formation |
|---|---|---|---|
| 680 | 715 | 35 | Anhydrite containing a little sand. |
| 715 | 735 | 20 | Anhydrite containing considerable calcareous material and some sand. On heating, a faint odor of bitumen was given off and a little sulphur deposited in the tube. Label says "gas." |
| 735 | 740 | 5 | Gray sandy limestone containing pyrite, and fragments of pebbles of gray and dark chert, and gypsum. Endothyra noted. Gave off odor of bitumen on heating. |
| 745 | 757 | 12 | Gray limestone containing sand most of which is below ⅛ mm. Chert, anhydrite, and pyrite. On heating a little oil distilled off. |
| 757 | 795 | 38 | Light gray dolomitic limestone containing sand, fragments of chert pebbles, pyrite and anhydrite. Gave off odor of bitumen on being heated. Little gas at 786. At 791 a little water, strong with sulphur. |
| 795 | 800 | 5 | Gray anhydrite, and quartz sand, some calcareous material and some dark chert grains. |
| 800 | 810 | 10 | Anhydrite, white, gray and yellow. |
| 810 | 822 | 12 | Anhydrite and very dark limestone. |
| 822 | 828 | 6 | Mostly white and gray anhydrite. Some very dark limestone, some sand and some fragments of greenish shaly rock. |
| 828 | 837 | 9 | Yellowish and dark limestone with sand and very fine gravel. Pyrite and anhydrite noted. Some foraminifera like a Textularia with perforate walls was observed, and an Anomalina, well preserved. |
| 838 | 843 | 5 | Limestone and anhydrite, both in very dark and in very light gray shades. Some sand present. Foraminifera of the types of Globigerina and Textularia quite common. |
| 843 | 857 | 14 | Anhydrite, mostly white, and some dolomite. Globigerina, Textularia and a foraminifera like Anomalina noted. Pyrite noted. |
| 857 | 900–943 | 83 | Light gray limestone containing chert, sand pyrite and considerable anhydrite. Gave off the odor of bitumen on being heated. Endothyra noted. Label said "little gas and oil." |
| 940–943 | 1,030 | 90 | Anhydrite containing calcareous material and a little sand. Some sulphur was deposited in the tube on heating. |
| 1,030 | 1,152–1,154 | 90 | Light gray limestone containing anhydrite and pyrite. Upon being heated, a little sulphur was deposited in the tube, and a faint odor of bitumen was given off. An Endothyra was noted. Label said "gas and a little oil." |
| 1,152–1,154 | 1,942 | 790 | Gray and dolomitic limestone containing anhydrite, pyrite, sand and a black mineral, probably chert, considerable oil was distilled off in heating the material. |
| 1,942 | 1,956 | 14 | Gray cherty limestone containing sand and anhydrite. Some oil was distilled off on heating the material. |

| From | To | Feet | Formation |
|---|---|---|---|
| 1,956 | 1,962 | 6 | Gray argillaceous dolomitic sandstone containing a considerable quantity of black mineral, probably chert. Most of the sand is below ⅛ mm. in size. Considerable oil was distilled off on heating the material. |
| 1,962 | 1,982 | 20 | Gray dolomitic limestone containing black fragments (probably carbonaceous material), pyrite, sand, chert and anhydrite. A little oil distilled off on heating. |
| 1,982 | 1,988 | 6 | Gray sandy marl, containing pyrite and black mineral. Considerable oil was distilled off on heating. |
| 1,988 | 1,994 | 6 | Gray limestone containing considerable chert and sand Some oil distilled off on heating. |
| 1,994 | 2,006 | 12 | Dolomitic sandstone containing pyrite and a black mineral. Most of the sand is below ⅛ mm. Gave off a little oil on heating. |
| 2,006 | 2,010 | 4 | Sandy dolomite containing pyrite. Considerable oil distilled off on heating. |
| 2,010 | 2,018 | 8 | Sand and clay (oil sand) containing a black mineral (probably chert). A little pyrite, a very little calcareous material. Gave off considerable oil on heating. Most of the sand is below ⅛ mm. Endothyra was noted (?). |
| 2,018 | 2,020 | 2 | Several barrels of oil were baled at 2020. |
| 2,020 | 2,028 | 8 | Brown limestone containing pyrite, anhydrite and sand. Some sulphur was deposited in the tube on heating. |
| 2,028 | 2,035 | 7 | Brown limestone containing considerable sand, most of it below ⅛ mm. and a little chert. On heating it gave off the odor of bitumen. An Endothyra was noted. |
| 2,035 | 2,041 | 6 | Argillaceous dolomitic sandstone containing anhydrite and pyrite. The majority of the sand grains are smaller than ⅛ mm. Some oil was distilled off on heating. |
| 2,041 | 2,045 | 4 | Gray argillaceous dolomitic sandstone containing a black mineral, probably chert. Some oil was given off on heating. Water was encountered at 2045. |
| 2,045 | 2,052 | 7 | Gray sandy dolomitic limestone containing a little anhydrite, pyrite and bitumen. A little oil was distilled off on heating. |
| 2,052 | 2,138 | 86 | Gray sandy dolomitic limestone containing pyrite, chert and a little anhydrite. Gave off odor of bitumen on heating. |
| 2,138 | 2,190 | 52 | Gray sandy limestone. Most of the sand is below ⅛ mm. Some oil distilled off on heating. |
| 2,190 | 2,197 | 7 | Blue-gray sandy dolomitic limestone. A little oil distilled off on heating. |
| 2,197 | 2,204 | 7 | Gray limestone containing some sand, most of which is below ⅛ mm. Some oil was distilled on heating. |
| 2,204 | 2,230 | 26 | Gray sandy dolomitic limestone containing a little chert (?). Gave off a slight odor of bitumen on heating. |
| 2,230 | 2,236 | 6 | Gray dolomitic limestone containing considerable sand, most of which is below ⅛ mm., and black fragmentary mineral. |

## Topography

No topographic survey of this area has been made. As already observed, the main topographic feature of this entire region is the Guadalupe Mountains and their southward extension into the Delaware Mountains. The elevation of El Capitan Peak, Guadalupe Mountains, is taken at 8690 ft. (2648 m.) and this is probably the highest point in the State. From these mountains the eastward slope is about 100 ft. (30 m.) to the mile to the Pecos River, a distance of about 60 miles (96 km.). The Rustler Hills, somewhat nearer to the river than to the mountains, intervene between the river and the mountains. They extend in a general northeast-southwest direction for more than 40 miles and form the western boundary of the Maverick Springs district. Their maximum elevation is about 4000 ft. (1219 m.). At intervals they are cut through by east and northeast drainage lines, such as Cottonwood Draw, Rustler Draw, Horseshoe Draw, etc. Although the mean annual rainfall does not exceed 10 in. (254 mm.) a great deal of water passes down these Draws and there has been much erosion.

In the Maverick Springs district, the principal sulphur deposits now exposed are in a valley known as the Virginia Draw, lying on the east and northeast side of the Rustler Hills and extending from Little Rock Tank, in Section 30, Block 111, to and into Section 24, Block 113, a distance of about 7 miles in a northeast direction. This valley is bounded on the west and northwest by the Rustler Hills and on the east and southeast by a low range of hills characterized by rounded knolls of dolomite.

The drainage down this valley is, thus, from the Rustler Hills and the lower dolomite hills towards the northeast. In ordinary times there is no surface water in this valley at all, but after the infrequent rains there are numerous natural tanks and a few rock tinajas in which water accumulates and lasts for a while. The slope of the valley is about 30 ft. to the mile and the throw of the drainage is against the dolomite hills in the upper part and against the outcrops of massive gypsum and gypsite in the lower part. Northeast of the Dot property, the almost flat part of the valley widens out to as much as 1200 ft. (366 m.) but is much indented, the low escarpment on the west encroaching on the flat and then withdrawing, forming small embayments.

A good deal of underground water finds its way down this Draw and it is always accompanied by more or less hydrogen sulphide. This water comes to the surface in the lower part of the Draw, just before its junction with Salt Draw, which latter Draw takes the drainage from Rustler Draw, Horseshoe Draw, Horseshoe Springs, etc. On the west side of Virginia Draw there is a low gypsum mesa, with sporadic outcrops of dolomite, and sulphur deposits have been found on this mesa especially near the Dot property, at depths as low as 50 ft. (15 m.).

## Water Supply of the District

The climate of this part of Texas is arid, the mean annual rainfall being between 8 and 10 in. (203 to 254 mm.). For long intervals there is no rain at all. During the winter months there may be a little snow or sleet, but it does not remain on the ground.

The shallow wells throughout the country yield an abundance of water, but it is highly impregnated with mineral matter. In Table 1 are collected analyses of water from a number of different sources. In regard to sample No. 2, Richardson says:

"Delaware Springs are at the head of flowing water in Delaware creek. Here water issues along the north side of the creek at the contact of gravel and a sandstone member of the Delaware Mountain formation. The quality of the water from all of the above springs is good and the water is soft. * * * But besides this good water several saline springs issue from the bed of Delaware creek in close proximity to the springs just mentioned having an altogether different and a deeper-seated source. The water in them bubbles up through joints in the Delaware Mountain formation that outcrops in the bed of the creek. This water contains considerable hydrogen sulphide, from which finely divided white sulphur is precipitated on contact with the air."

Dr. Booth mentions the presence of sodium sulphide in the water from a spring at the head of Delaware Creek and also in the water of the creek 30 miles below the springs and 10 miles from the mouth of the creek, where it empties into the Pecos River. He also found this ingredient in the water from Mustang Springs, Delaware Mountains. There can be no doubt of the existence of alkaline sulphides in the waters of the sulphur district and this fact has an important bearing on the origin of the sulphur deposits, as was observed by Blake more than 60 years ago.

Samples 6 and 7 taken in 1904 were from the same waters in Salt Draw as samples 8 and 9, but they were secured in February after a long period in which there was no rain. Samples 8 and 9 were taken in November after the fall rains and they show a lower concentration than the others. The absence of hydrogen sulphide from the running water is due to the oxidation of this compound. Free sulphur is now being deposited from these waters, especially at the Falls. Sample 8, from the pool at the head of water in Salt Draw, represents the underground drainage that comes into this Draw from Rustler Draw and Horseshoe Draw. Sample 9, of running water from Salt Draw, represents the water after it has been mixed with the drainage from Virginia Draw. On the whole these waters are much alike. Sample 12, from Maverick Springs, is closely similar in composition to sample 10, obtained from a depth of 23 ft. (7 m.) in the Virginia Draw. The locality in this Draw is 2 miles (3.3 km.) southwest from Maverick Springs and is separated from Maverick

TABLE 1.—*Analyses of Surface Waters (Grains per U. S. Gallon)*

| Sample No | Ca | $CaSO_4$ | Mg | $MgSO_4$ | MgC | Na+K (as Na) | NaCl | $Na_2CO_3$ | $Na_2SO_4$ | $Al_2O_3$ | As | $SiO_2$ | $SO_4$ | $CO_3$[7] | Cl | S (as $H_2S$) | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | . | 218 26 | ...... | 11.25 | ....... | ....... | 23 10 | 4 83 | ... | Tr[1] | .... | 3.44 | ... .. | .... | .... | .. | 260 88 |
| 2 | 6 64[9] | ..... | 5.89 | ........ | ....... | .... | ........ | ........ | ...... | 0 32[1] | .. | 0 28 | 2 46[2] | .... | .... | ... | 184 53 |
| 3 | 9 351[10] | 111.790 | .... | ........ | 6.337[11] | ...... | 20 364 | ........ | .... | Tr[1] | . | 1.586 | .. | .. | . .. | . | 183 627 |
| 4 | ..... | 105 53 | ..... | 11.26 | ........ | ....... | 6.82 | ........ | 53 92 | 22.73[3] | Tr | 12.82 | ...... | . | . | . | 213 08 |
| 5 | 3 00[5] | 90.36 | ..... | 5 03 | ..... . | ....... | ......... | ........ | 61 55 | 1,291.92[4] | Tr | ...... | . | ... | . .. | ..... | 1,451.86 |
| 6 | .. ..... | 274.087 | ...... | 72.128 | 42 127 | ....... | 1,026.181 | 73.784 | .... . | 0.839[6] | .. | 0 046 | . ... | . . | . | . .... . | 1,489.192 |
| 7 | ........ | 234.202 | ...... | 124.737 | 32 578 | ... | 1,309.011 | 126.493 | ....... | 1.445[6] | .... | 2.705 | . . | . | ..... | . .... | 1,831.202 |
| 8 | 57 066 | ....... | 15.192 | ....... | ...... . | 118 159 | ......... | ........ | ...... | ....... | .. | ..... | 154 336 | 6 185 | 209 477 | 1.967 | |
| 9 | 56 833 | ....... | 9.803 | ....... | ........ | 147.626 | ......... | ....... . | ... . | . . . | . | ... | 149 551 | 6.827 | 242.153 | None | 612.792 |
| 10 | 80 756 | . ..... | 22 408 | ...... | ........ | 498.192 | ..... ... | ..... . | ... .. | ....... | .. | . .. | 226 748 | 17 913 | 794 536 | ....... | 1,640.553 |
| 11 | 73 696 | ... ... | 17.681 | ....... | .... ... | 349.517 | ......... | ........ | . .... | . . ... | . | . .. | 177.326 | 26.316 | 573 294 | 5.358 | 1,225 198 |
| 12 | 66.986 | . ..... | 35.243 | ....... | .. . .. | 498.309 | ......... | ........ | ....... | ......... | .... | ... . | 231 066 | 23 223 | 742 053 | 3 793 | 1,600.473 |

Description of Sample

Sample No. 1—From the Cooksey ranch, Section 15, Block 114. Richardson.

Sample No 2—From a spring at the head of Delaware Creek. Richardson.

Sample No. 3—From "Stinking Seep," Section 9, Block 114, northwest of Cooksey ranch. Analysis by S. H. Worrell of sample by the writer, February, 1904. This water had a nauseating taste and carried considerable hydrogen sulphide. It had the local reputation of being poisonous, and at the time of the writer's visit, there were many bones of cattle, birds, etc., to be seen.

Sample No. 4.—From Section 11, Block 114, 25 ft below surface. E. M. Skeats. This water contained 79 08 grains of free sulphuric acid per gallon.

Sample No. 5.—Sample from the Dot property, in Virginia Draw, 10 ft. below the surface, carried still more free acid.

Sample No 6—From Salt Draw, north of house at Maverick Springs. Analysis by S. H. Worrell from sample by the writer, February, 1904.

Sample No. 7—Same, west of house. This sample contained much free hydrogen sulphide.

Sample No. 8.—From pool at head of water, Salt Draw. Analysis by W. T. Read from sample by the writer, Nov, 1916.

Sample No 9—Same from running water below the falls.

Sample No. 10—From Virginia Draw, northwest part of Section 1, Block 111, 1½ miles below the Dot property, at a depth of 23 ft. Analysis by W. T. Read, from sample by the writer. This sample smelled strongly of hydrogen sulphide, but was exposed to the air and the gas escaped, so that it does not appear in the analysis.

Sample No 11—From the Big Hole in Virginia Draw, near road from Maverick Springs to house of W. H. Hicks; 1 mile below the location of sample No. 10. Analysis by W. T. Read from sample by the writer

Sample No. 12.—From seepage in a shallow pit at Maverick Springs, Section 18, Block 45 Analysis by W. T Read from sample by the writer

[1] Alumina and iron.
[2] Combined sulphuric acid.
[3] $Al_2(SO_4)_3+Fe_2(SO_4)_3$
[4] $Al_2(SO_4)_3$, 628.47; $Fe_2(SO_4)_3$, 663.45.
[5] Phosphate of lime.
[6] This carries a trace of iron.
[7] Bicarbonate.
[8] MgO.
[9] CaO.
[10] CaS.
[11] Also 34.199 $CaCl_2$.

Springs by the ridge that lies between Virginia Draw and Maverick Draw. As the water in the Virginia Draw is probably drainage from the sulphur deposits higher up the valley, so the water in the "seep" at Maverick Springs may be the drainage from the sulphur deposits that are west of the Springs.

There is abundance of ordinary "gyp" water in Maverick Springs at all times. It is the largest and most permanent water supply in the district. Above the Springs there is a large drainage area extending to and beyond Big Rock Tank, Section 26, Block 111. There are no known sulphur deposits in Maverick Draw and the water is as good as any in the country.

By constructing an earth dam below Maverick Springs, it would be possible to impound a large quantity of water, and it is thought that this will have to be done, if the district is developed. Of course, there is more or less water to be obtained from wells sunk in the Maverick Draw above the Springs.

The Michigan Sulphur & Oil Co. is now obtaining its water from a windmill in Rustler Draw, about 9000 ft. from the property, through a gravity pipe line. The supply is sufficient only for a small camp and a limited business.

It is uncertain whether artesian water suitable for use could be secured within reasonable drilling depths. For a part of the year, the time depending on the rains, "soft" water can be obtained from natural water holes, but this source of supply is too uncertain to be considered seriously.

### Fuel

The local fuel is juniper, or mountain (scrub) cedar. It is hauled in from points 5 to 10 miles distant (8 to 16 km.). It makes a quick and hot fire, but has no lasting qualities. The supply of this fuel is sufficient only for ordinary household purposes, cooking, heating, etc. It cannot be depended on for raising steam in a plant of any size.

Twenty miles (32 km.) southeast of the Maverick Springs district, in Section 16, Block 59, Reeves county, are the shallow oil wells on the Huling-White-Ross ranch. They have been known for some years, but have not been used as a source of oil except for lubricating windmills and so forth. The oil is an excellent natural lubricant and comes from a depth of about 230 ft. (70 m.).

Fuel for industrial purposes in this district will have to be brought in over a branch line of railroad built from Orla, 15 miles from Maverick Springs. Best screened Texas lump coal can be laid down at Orla for $5.15, the haul being 366 miles from Thurber.

There has been more or less talk about the natural gas in the district being sufficient for industrial purposes. There is little except hope on which to base such expectations. The gas that has been found there

comes from shallow depths and no one competent to judge has so far expressed a favorable opinion on the possibilities of finding enough of it, in depth, to be of consequence. There is many a wish that is father to the thought.

## Nature and Origin of the Deposits

Skeats, in the bulletin previously referred to, describes at considerable length the topography and geology of the district.

A test pit near Maverick Springs, in a sulphur deposit of about 80 acres in extent, gave the following results:

*Section of Sulphur Pit, Maverick Springs District*

| | Feet | Inches |
|---|---|---|
| Earth | 1 | |
| Gypseous sand | 1 | |
| White gypsum | 3 | |
| Gypsum, with 4 per cent. of sulphur | 1 | 6 |
| A hard shaly and gypseous formation, with gravel, sulphur 31 per cent. | 4 | 6 |
| Sulphur ore, with 44 per cent. of sulphur | 1 | |
| Light brown gypseous material, with 29 8 per cent. of sulphur | 2 | |
| Light brown gypseous material, with 31.1 per cent. of sulphur | 2 | |
| Soft white material, with 12 9 per cent. of sulphur | 2 | |
| Soft white material, with 7 1 per cent. of sulphur | 2 | |
| Soft white material, with 5.5 per cent. of sulphur | 2 | |
| Black gravel with gypsum crystals and 11.9 per cent. of sulphur | 4 | |
| Black gravel with gypsum crystals and 35 3 per cent of sulphur | 2 | |
| Black gravel with gypsum crystals and 36 5 per cent. of sulphur | 2 | |
| Black gravel with gypsum crystals and 21.4 per cent of sulphur | 2 | |
| Blue ore with 46 per cent of sulphur, in streaks | 11 | |
| Left off in sulphur ore. | | |

Disregarding the 6½ ft. of earth, gypseous sand and white material just below the surface, the 36½ ft. to the bottom of the pit, 43 ft., averages 29.50 per cent. of sulphur.

His observations on the origin of the sulphur deposits are particularly pertinent, and agree in great measure with those of William P. Blake, of the Pope Expedition, and with those of later geologists and engineers. In this connection he says:

"The origin of the sulphur is not easy to decide, for while it is generally thought to be of solfataric origin yet there is little evidence to support this opinion. All of the sulphur of the region occurs in and with gypsum and in connection with water containing sulphuretted hydrogen. This water has been traced by the writer from the upper end of Crow Flat, a valley on the east side of the Guadalupe Mountains.

"If decomposition of the sulphuretted hydrogen produced the sulphur, the deposits should occur over a wider area than is now thought to be the case. Sulphur may be produced by the alteration of gypsum, but in this case we would have to consider the nature of the reducing agent and explain why its action appears to be confined to a line near which passes an underground stream of sulphur water. From a careful study of the country and the processes now at work there, the writer is of the opinion that the richer bluish ores have been formed from sulphur waters at a time when they were above ground, and probably through the agency of certain algæ which

are plentiful in the sulphur springs today. The organic matter associated with these ores gives strength to this view, and also the fact that it is quite common to find the black gravel composed of small rounded pebbles in alternation with the ore. The ores in which the sulphur occurs as crystals are probably formed by the decomposition of sulphuretted hydrogen given off from the highly charged water when it enters a porous or broken stratum.

"In some places the ore may be a replacement product, the original gypsum having been removed. It is possible, also, that what is now gypsum was once carbonate of lime. Instances might be given of the occurrence of gypsum hills capped with limestone, or dolomite, which is in close association with underlying beds of gypsum. In many places it is difficult to say just where the limestone ends and the gypsum begins.

"These are points of great practical importance, for if we knew the mode of formation of the sulphur we would know where to look for it, and could form some idea of how much to expect. Only these deposits which have been uncovered by erosion have as yet been found, and it is likely that systematic prospecting would bring to light other deposits in the region."

In 1891, O. C. D. Ross presented a paper before the British Association for the Advancement of Science on the origin of sulphur, petroleum, etc.[10] Mr. Ross gave a series of hypothetical chemical reactions to show that carbonate of lime, sulphuretted hydrogen and peroxide of hydrogen could yield gypsum and hydrocarbons analogous to ethylene ($C_2H_4$). Mr. Ross's theory is ingenious but does not rest upon any known facts.

The possible influence of sulphur-secreting algæ on the origin of sulphur is also interesting. The Beggiatoa, or sulphur-bacteria, contain grains of sulphur and these may have arisen through the oxidation of sulphuretted hydrogen in a manner comparable with the power of Crenothrix and Leptothrix ochracea (iron bacteria) to oxidize ferrous compounds to ferric compounds. The disappearance of the sulphur grains in the Beggiatoa has been thought to be due to their oxidation to sulphuric acid.

With reference to the influence of certain algæ, mentioned by Skeats as occurring in the sulphur waters today, Dr. Emile Böse, of the Bureau of Economic Geology and Technology, University of Texas, suggested that the small holes in the magnesian limestone of the Rustler formation may be the remains of groups of algæ. These markings, mentioned by Richardson, on page 44 of his report, occur so plentifully as to suggest a vigorous and continuous growth. They vary in diameter from a fraction of an inch to more than an inch and are of circular cross-section. For the most part they are now well crystallized calcite. They occur in great abundance on a small dolomite hill near the center of Section 1, Block 111, as also in slabs scattered on the surface of the mesa between Virginia Draw and Lindsey's house. Richardson speaks of them as occurring in a section of 50 ft. (15 m.) in Horsehoe Draw, associated with

[10]An abstract of the original paper appeared in *Chemical News* (1891), **64**, 191, and Mr. Boverton Redwood commented on it in the same volume, page 215.

a calcareous buff sandstone and Castile gypsum. Bischof[11] gives many instances of the transformation of limestone into gypsum through the action of sulphuretted hydrogen (hydrogen sulphide). Thus, Covelli found beautiful crystals of gypsum and sulphur in the crater of Vesuvius. In examining the fumaroles of Tuscany, Dumas found no free acid and he ascribed the alteration of the limestone there into gypsum to the action of sulphuretted hydrogen and connects this with a similar phenomenon at the sulphur baths, near Aix, where the walls of limestone gradually become covered with crystals of gypsum. Hoffman found extensive deposits of gypsum in the Lipari Islands, formed through the action of sulphuretted hydrogen on limestone, and gives an instance of the occurrence of gypsum in tuff beds underlying lava. Not only are limestones thus decomposed, but even silicates. There is abundant evidence of the formation of gypsum and sulphur through the action of sulphuretted hydrogen on limestone. Bischof says:

> "The simultaneous formation of gypsum and separation of sulphur may be effected artificially by bringing sulphuretted hydrogen in contact with moist chalk. When this gas is heated up to 212° F., sulphur is deposited in the conducting tube, where it comes in contact with the air; it may, therefore, be readily understood that moderately heated sulphuretted hydrogen, continually streaming into a bed of limestone, may give rise to the formation of gypsum and the deposition of sulphur, when atmospheric air is not excluded. While, on the one hand, sulphur is separated from gypsum which is saturated with bituminous substances by a simple decomposition, sulphuretted hydrogen streaming through beds of limestone gives rise to the simultaneous production of gypsum and sulphur; the very association of these two substances admits of no other explanation."

It would appear that the sulphur deposits in Culberson county may have arisen in one or more of the following ways:

1. Through the action of hydrogen sulphide on limestone or dolomite.
2. Through the decomposition of waters carrying alkaline sulphides.
3. Through the decomposition of gypsum impregnated with bituminous substances.
4. Through the precipitation of sulphur from sulphur-bearing gases, especially sulphuretted hydrogen.

The conditions under which these factors have operated are observable in the district today, but they may have been more active and intensive than they are now.

It is not probable that volcanic or seismic agencies are responsible for the deposition of this sulphur, although they may have been influential in causing or accelerating dislocations or alterations of the strata and may thus have provided for the circulation of sulphur-bearing waters and gases. It is possible that the alteration of gypsum impregnated with bituminous substances contributed to the formation of some of the deposits, for there are many localities where bituminous gypsum is

[11] *Chemical and Physical Geology.*

observable in close association with sulphur. This is particularly the case at the Grant mines, Sections 17 and 18, Block 61, and on Section 1, Block 46, where there are heavy beds of a wavy gypsum carrying a dark brown bituminous substance.

## Utilization of the Deposits

So many factors enter into the utilization of these sulphur deposits that to discuss them in detail would protract this paper to an undue length. The principal ones are:

The nature and thickness of the overburden.

The nature and thickness of the several bands of sulphur-bearing material that are within workable distances from the surface.

The methods of extraction.

Transportation of the product to available markets.

Sale of the product, inclusive of the capital required for carrying stocks.

The development work that has been done is not sufficient in character or extent to allow one to answer any one of these questions as fully as could be desired. While the writer believes that some of the deposits are well within commercial possibilities, this belief is based more upon his knowledge of such deposits in general than upon the results of detailed prospecting and development on any particular area. In the district as a whole the work already performed has not been conducted systematically. The statements of millions of tons of workable material have not been confirmed by regular sampling and analysis. Entirely too much has been assumed, both with respect to the thickness of the several strata carrying sulphur and the composition of these strata. The writer has before him now a statement that on such and such a holding there are more than one million tons of 99.85 per cent. pure sulphur and that a considerable acreage represents more than two million tons of pure sulphur. Statements of a like nature with respect to other properties have a similar basis of hope. It is idle to consider such reports as forming a solid foundation on which actual development work may be undertaken. With the best intentions, no engineer or geologist who is careful of his reputation can afford to certify to anything more than has been determined with reasonable accuracy. In deposits of this character, subject to considerable variations in horizontal extent, depth and content of recoverable values, there is nearly always a margin of risk which must be taken into consideration. It is the purpose of systematic prospecting to reduce this risk to a minimum, but it can never be wholly removed, and the degree of the removal is not always determinable.

The advantages that this sulphur district may have are: (1) it is 700 miles nearer the western points of actual and possible consumption

than the deposits in Texas at the mouth of the Brazos River; and (2) the initial investment would be comparatively small with respect both to prospecting and development.

No reliable data can now be given on the cost of production. The operations have been conducted on so small a scale and so intermittently that the cost accounts that have been kept are not reliable. The same observation applies, on the whole, to the quality of the material obtainable. Instead of a scant 20 or 30 analyses there should be hundreds, following careful sampling. Whether the available material carries 25 or 30 or 35 per cent. of sulphur is not known. Picked specimens of 50, 60 and even 90 per cent. sulphur cut no figure, except a delusive one. They do not mean anything to the engineer engaged in a critical examination of actual conditions. Such hand specimens are nice to have around and to send to museums, etc., but when it comes to real business they are very good things to let alone.

In summing up this paper, I wish to say that some of the sulphur properties at and near Maverick Springs and towards the upper part of Virginia Draw seem to be worthy of systematic prospecting. There is a considerable amount of sulphur in this district within 50 ft. (15 m.) of the surface and it can be mined cheaply. I am disposed to take the average content of sulphur in the crude material as not exceeding 30 per cent., although there are strata of higher sulphur content. That is to say, the average content of native sulphur in the material that can be mined and sent to the extractor will be around 30 per cent. There will doubtless be strata of a higher content and strata of a lower content.

It is not known today how much of the area involved will yield material carrying 30 per cent. of sulphur, nor what is the average thickness of the overburden that will have to be removed.

I take it that practically all of the work in mining the sulphur will be by the open-pit and bench system, the overburden being removed and sent back to allow of the full lift of the shovel. This system could be used to a depth of 50 ft.

Underground operations would involve a considerable expense for timbering and ventilation, and are not now to be seriously considered. Looking at the matter as a whole, I do not think that plans for actual development can now be made owing to the lack of systematic prospecting, and it is too early in the game to plan for extraction plants. Such a plant is the last thing to be taken up. Its location, design and equipment depend on factors not yet clearly understood. It requires a good deal of money to go into the sulphur business and it requires a good deal to stay in it, to be ready to take contracts and to fill them promptly. At the same time, and knowing the situation in Culberson County thoroughly, I am of the opinion that some of the sulphur deposits there are worthy of attention as a continuous source of sulphur of good quality.

## Discussion

S. J. Jennings, New York, N. Y.—If there is any large quantity of material which will yield 25 per cent. of sulphur, which is worth, at the present time, about $7.50 a ton in the ground, capital would undoubtedly be rapidly attracted to the deposit. The question to be decided is whether there is a sufficient quantity to warrant a railroad and the building of a plant to recover the sulphur.

W. B. Phillips.—While I believe, from what I have seen there during a recent two months' visit, that some of the deposits can be worked profitably, up to this time not one of them has been sufficiently developed. The reason is that the present owners, who obtained title partly from the State and partly by purchase from individuals, have such extravagant ideas of their value that they are asking prices which absolutely prohibit any company from beginning development at the present time. For instance, one of the companies asked $80,000 for an option on 300 acres of undeveloped land. Another adjoining property is held at $600,000, undeveloped, and a third at $500,000. I believe, from an engineering and economic standpoint, that some of those deposits can be profitably worked and that they will be worked when the present owners get down to business.

E. T. Lednum, Joplin, Mo.—Is not most of that land school land, and, under the Texas mining law, would it not be possible for anybody to enter and operate property adjoining that on which you are supposed to hold a lease?

W. B. Phillips.—The Texas Legislature enacted a new mining law last winter. As it stands now, you can take up as much as four sections of the sulphur land, about 2400 acres, on which you must do the usual assessment work, and pay a royalty to the State of 25 c. a ton for such stuff as is produced and sold. The country has already been thickly covered with claims.

L. W. Trumbull, Cheyenne, Wyo.—I am interested in Dr. Phillips' description of the probable mode of formation of sulphur deposits in Texas, because we find almost identical conditions in Wyoming, where deposits of brimstone underlie gypsum beds which lie above the Carboniferous limestones. His theory regarding their formation is very similar to the one we have arrived at in our State.

We have been producing sulphur spasmodically for 10 or 12 years, but for some reason we cannot market the material when the price goes below $22 a ton in spite of the fact that from many of the chambers the brimstone runs up to 95 per cent. sulphur. The deposits are kidney-shaped, some of them having greater vertical extent than horizontal, the stratification dipping at 18 or 20°. They are on the northern side of Owl Creek mountains, close to railroads.

# Ore Deposits of the Boulder Batholith of Montana

## A Genetic Description

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(St. Louis Meeting, October, 1917)

### OUTLINE



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*Granite Period.*

A. Granite Phase.
  1. General Description.
  2. Segregations.
    (*a*) Pyrrhotite—Spring Hill Mine.
    (*b*) Magnetite—Golden Curry Mine.
    (*c*) Quartz—Quartz Mass.
  3. Disseminations.
    (*a*) Auriferous Pyrite—Red Rock Creek.
    (*b*) Cupriferous Pyrite—Heddleston.
    (*c*) Enriched by Hydrothermal Solutions—Golden Sunlight Mine.
  4. Contact Deposits.
    (*a*) Contact-Metamorphic—Cable.
    (*b*) Replacement—Southern Cross.
    (*c*) Fissure—Gold Coin.
  5. Fissure Veins.
    (*a*) Closely Following Intrusion—Gold Quartz, Pyrite Veins—Garnet District.
    (*b*) Later in Age—Gold Quartz, Calcite Veins—Marysville District.
  6. Summary.

B. Aplite Phase.
  1. General Description.
  2. Segregations.
    (*a*) Pegmatite—Quartz—Lost Creek.
    (*b*) Tourmaline—Valley Forge Mine, Rimini.
    (*c*) Quartz—Quartz Mass—Basin.
  3. Disseminations.
    (*a*) Chalcopyrite—Modoc Mine, Red Lion District.
  4. Fissure Veins.
    (*a*) Origin.
    (*b*) Distribution.
    (*c*) Character.
    (*d*) Geographic Variation.
    (*e*) Recapitulation.

C. Quartz-Porphyry Phase.
  1. General Description.
  2. Disseminations.
    (*a*) Cupriferous Pyrite—Tropic Mine, Butte.
  3. Fissure and Fault Veins, Butte District.
    1. General Description.
    2. East and West Veins.
    3. Northwest Fault Veins.
    4. Northeast Fault Veins.
  4. Summary.

*Rhyolite Period.*

A. Early Rhyolite Phase.
  1. General Description.
  2. Dissemination—Porphyry Dike.
  3. Impregnations—Woodrow Wilson.
  4. Contact Deposits—Monte Cristo.

B. Dacite (or Later Rhyolite) Phase.
  1. General Description.
  2. Fissure Veins—Ruby.

C. Summary.

PART 3.—CONCLUSIONS



## INTRODUCTION

THE purpose of this paper is a comparison, based on genesis, of the ore deposits associated with the igneous rocks of the central Montana Rockies. Considered separately, without attention to their origin, the orebodies appear to be distributed haphazard, with no fundamental laws controlling their position and character. It is believed, however, that if all the factors be weighed, this apparent confusion will to a large degree disappear. The first essential to such a comparison is detailed knowledge of the individual mines, as to their geographic position (see Fig. 1), their mineralogical character, and in particular the derivation of their ores.[1]

The authors of this paper have pursued such an investigation for the past 6 years, and have accumulated a mass of evidence that enables a preliminary correlation to be attempted. It appears that the majority of ore deposits are genetically related to one or another of the igneous

[1] In the collection of evidence for this paper the authors have personally examined a large proportion of the ore deposits of the Boulder Batholith. Published descriptions, however, particularly those of the U. S. Geological Survey, have been used, credit being given in each instance. Particular mention can hardly be made of the universal kindly assistance afforded the authors in their quest for material for this paper, but in no instance was such help refused, although in many cases it necessitated a considerable effort on the part of the donors, or the use of maps and other material not heretofore made public. Particular thanks for the contribution of facts and for criticism are due to Messrs. R. H. Sales, C. W. Goodale, F. A. Linforth, M. H. Gidel, and D. C. Bard. The great mass of detail which would be necessitated by the publication of all the supporting evidence, would, in the authors' opinion, obscure the more important points. Therefore, in so far as possible this evidence has been given graphically in the accompanying plates. It has been the endeavor of the authors to make these plates self-explanatory, in order that written descriptions of them might be omitted from the text. These plates are grouped at the end of the paper.

periods of the region; that when grouped according to this relationship they display resemblances and contrasts that can be traced with certainty to the conditions of their origin; and that common laws governing the derivation of their varying characters can be reasonably deduced. In short, a true genetic classification throws considerable light on the following subjects:

1. The comparison between igneous periods and their accompanying ore deposits.
2. The progression of type of ore deposits within a single igneous period.
3. The relation between the mineralizing activity and the form of igneous intrusions.
4. The geographic variation of vein-forming solutions of common origin.
5. The vertical distribution of sulphide minerals in veins formed by ascending solutions.

This paper, therefore, will summarize as briefly as possible the more significant features of the Montana mining districts, drawing from these such conclusions upon the natural history of magmatic ore deposits as appear inevitably to follow.

## PART 1.—GEOLOGIC BACKGROUND

### A. Introduction

From the earliest development of the mining industry in Montana, the close association of the ore deposits with rocks of granitic type has been observed.[2] In fact, the presence of "true fissures" in granite walls was eagerly sought, and where found was seized as indisputable evidence of the value and permanence of the prospect. The rise of many great mining camps has testified to the foundation of truth beneath this assumption of the prospector, and Bannack, Argenta, Unionville, Wickes, Marysville, and Butte, have successively found granitic rocks a congenial home for ores of the most diverse types.

More recently it has become rather generally accepted that the major portion of these granite outcrops represent a single great mass of plutonic rock and in many instances identity has been established by continuous surface exposure. To this great granitic mass the term Boulder Batholith, originally applied to a limited outcropping area, has been extended.

### B. General Geology

The Boulder Batholith represents the middle phase of an igneous cycle that during Cretaceous and Tertiary times was the dominating geologic event of the Montana Rocky Mountain region. The rock types of closely associated eruptions pass through a similar sequence; first,

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[2] See the volumes of *Mineral Resources of the Western States* for the years 1866 to 1885.

gabbro, diabase, andesite; second, diorite, quartz-monzonite, aplite (and quartz-porphyry); and third, dacite, rhyolite. These are generally grouped under the terms andesite, granite and rhyolite series, a succession which is found throughout southwestern Montana.

The several eruptions and intrusions are intimately related to the great mountain-making epochs of the Cordilleras, as may be seen by reference to the following chronological table[3] (see Fig. 23).

1. Middle Cretaceous. Period of main Rocky Mountain folding, and formation of large earth-folds in northwesterly direction.
2. Middle-Upper Cretaceous. Extensive erosion and beveling of folds.
3. Upper Cretaceous. Andesite eruption Deposition of extrusive lavas and breccias west of Rocky Mountain front and formation of tuffs and andesitic sediments on the plains.
4. Upper Cretaceous Local intense erosion and formation of coarse andesite conglomerates.
5. Upper Cretaceous. Thrust faulting along northwest lines, and local intensification of folding.
6. Eocene(?). Intrusion of Boulder Granite Batholith.
7. Eocene. Extensive erosion, approximating peneplanation.
8. Oligocene, Miocene. Normal faulting; accumulation of river gravels and lake silts; early rhyolite.
9. Pliocene. Same conditions, with extrusion of later rhyolite and dacite.
10. Pleistocene. Two or more glacial stages in the mountains.

It will be noticed that the major igneous periods—the andesite and granite—successively followed the epochs of more violent tectonic disturbance of the region. Thus the andesite eruption, involving the transfer of tremendous masses of rocks from the depths to the surface, succeeded the period of main Rocky Mountain folding, while the granite, even more extensive in its effects, followed the later state of earth-movement represented by the thrust faults. The conclusion that the igneous activity is thus in some way dependent upon the changing stresses and pressures of the mountain-making periods can scarcely be avoided. Further corroboration is afforded by the later rhyolite, which follows closely the period of normal faulting.

## C. Igneous Rocks

The Andesite stage of igneous activity is represented by a very diversified group of rocks: gabbro, basalt, diabase, augite, andesite, andesite porphyry with varying types of ground mass and phenocryst, amygdaloidal andesite, andesite breccia and tuffs are among the forms encountered.

These rocks were erupted from two main vents, one in the Crow

[3] Adapted from The Boulder Batholith of Montana by Paul Billingsley. *Trans.* (1915), **51**, 31.

Creek Mountains[4] near Elkhorn, and the other in the vicinity of Thunderbolt Mountain, east of Deer Lodge (see Fig. 2). In these regions the gabbro, in the form of dikes and intrusive masses, is most abundant; around the vents the andesite breccia is widely distributed, and radiating from these peaks the surface flows spread out in overlapping layers covering the many square miles of the andesite areas. In general, these series of surface flows pass from basalt below to andesite porphyry in the later lavas, and suggest an increasing acidity of magma throughout the period.

The granite stage followed the andesite after a considerable interval in which much erosion, and additional earth movements, occurred. The igneous rocks of this period are closely associated and follow each other with very slight breaks. A basic diorite, locally grading down into pyroxenite and pyrrhotite, is the earliest type. It is found in part as the basic contact of the batholith itself, and in part as inclusions, probably representing a deep-seated earlier chilling of the magma. This is followed by the main intrusive rock, a quartz-monzonite, variable in texture but essentially uniform in composition throughout the extent of the Boulder Batholith. This mass was intruded at a comparatively low temperature (500° C.) and reached its present position, high up in the sedimentary series, by the mechanical action of the magma in breaking off blocks of the roof and penetrating upward into the space thus vacated (magmatic stoping). In general the contact is more basic than the main mass, but locally siliceous stringers of the later segregations have pushed out beyond the borders. The crystallization of the quartz-monzonite magma was followed, while the rock was still warm and possibly viscous, by widespread local intrusions of a more siliceous residue. This forms the aplite, and comprises possibly 8 or 10 per cent. of the total mass. It abounds in pegmatitic segregations, which occasionally pass into nearly pure magmatic quartz. Pegmatite, quartz, and tourmaline, in varying proportions, appear to be essentially contemporaneous in the later aplite stages. Very rarely intrusions of quartz-porphyry are found. This rock is slightly more basic than the aplite, from which, however, it differs mainly in texture. The two are closely associated in the Butte mines, where the quartz-porphyry is best known. Cooler contact phenomena and subsequence to jointing common to both granite and aplite, place the quartz-porphyry in a later period, but chemical similarity and evidences of transition with depth suggest that the sources of the latter rock must be of aplite type and may be of identical origin with the aplite segregations of the upper horizons.

The final fractionalization of the granite period is represented by metalliferous quartz, which in its distribution closely follows the aplite

[4] R. W. Stone: Geological Relations of Ore Deposits in the Elkhorn Mountains, Montana, *U. S. Geological Survey, Bulletin* 470 (1911), 75.

and quartz-porphyry. This—the siliceous residue of a magma which successively crystallized as diorite, quartz-monzonite, and aplite—has been the source of the great ore deposits of the batholith.

Like the earlier andesite stage, the granite period was followed by an interval of geologic quiescence and erosion, during which the Eocene peneplain and the broad valley floors became established. Before the close of this period the early rhyolite was extruded. This rock, formed as intrusions, breccias, and surface flows, rests upon the partially levelled surface of the granite hills; but underlies the later Tertiary deposits of the valleys. There is much evidence, especially in the Rimini region, that the early rhyolite antedated the period of block faulting that initiated the upper Tertiary.

It is definitely known that the later rhyolite dacite series comes late in the Neocene, as its tuffs, with leaf impressions, are interbedded with Miocene and Pliocene gravels. The rhyolite-dacite is a rock of local distribution, with vents scattered broadcast around the area of the batholith. Breccia in the vent itself, radiating dikes, and a capping of extrusive material commonly characterize these local eruptions. Close association with the granite rocks suggests local reservoirs within the batholith as their sources.

Rare occurrences of late diabase of Pliocene age mark the last phase of igneous activity in the region.

Table 1, based mainly on analyses of the U. S. Geological Survey, will indicate the variation in composition of the several igneous stages, as well as the differentiation within each period.

TABLE 1

| | | $SiO_2$ Per Cent | $Al_2O_3$ Per Cent | $Fe_2O_3$ Per Cent | FeO Per Cent | MgO Per Cent | CaO Per Cent | $Na_2O$ Per Cent | $K_2O$ Per Cent |
|---|---|---|---|---|---|---|---|---|---|
| Andesite, 47–65 per cent. $SiO_2$ | Gabbro | 47.0 | 18.0 | 5.4 | 6.0 | 5.50 | 10.0 | 2 6 | 1 2 |
| | Augite-Andesite | 52.0 | 14.5 | 3.5 | 4.5 | 9.00 | 5.5 | 3.0 | 3.9 |
| | Hornblende-Andesite | 65 2 | 16.6 | 3.3 | 3.5 | 3.40 | 7.1 | 2.4 | 2.8 |
| | Latite | 64.5 | 17.7 | 1.3 | 1.9 | 0.60 | 3.7 | 3.8 | 3.7 |
| Granite 59–77 per cent. $SiO_2$ | Pyroxenite | 51.8 | 8.0 | 1.5 | 8.3 | 24.10 | 5 3 | 0.4 | 0.1 |
| | Diorite | 59.8 | 16.7 | 2.3 | 3.4 | 3.20 | 6.1 | 3.1 | 3.3 |
| | Quartz-monzonite | 64.6 | 15.4 | 2.0 | 2.5 | 2.00 | 4.0 | 3.0 | 4.1 |
| | Aplite | 76.2 | 12.9 | 0.7 | 0.1 | 0.20 | 0.9 | 2.6 | 5.6 |
| | Quartz-porphyry | 69.9 | 15.1 | 0.4 | 0.8 | 0.60 | 1.5 | 2.7 | 6.4 |
| Rhyolite, 67–75 per cent. $SiO_2$ | Early rhyolite | 75.3 | 11.9 | 2.2 | N.D. | 0.10 | 0.6 | 3.1 | 5.0 |
| | Dacite | 70.9 | 14.3 | 0.9 | 0.8 | 1.00 | 1.0 | 3.3 | 3.8 |
| | Diabase | No analysis available. | | | | | | | |

## D. Classification of Ore Deposits[5]

Each period of igneous activity stimulated the formation of ore deposits, the common succession being, roughly, segregations and disseminations within the magma, contact deposits in the intruded rock, fissure veins in the igneous rock, and later mineralized faults. A genetic classification will therefore consider this order, together with the cycle of igneous rocks. The following is adopted here in preference to a system based on form of deposit or nature of vein filling.

1. *Andesite Period.*
  1. Andesite Phase.
    A. Disseminations.
      1. Amygdaloids.
      2. General Impregnation.
    B. Contact Deposits.
    C. Fissures.
2. *Granite Period.*
  1. Granite Phase.
    A. Segregations.
      1. Magnetite.
      2. Pyrrhotite.
      3. Quartz.
    B. Disseminations.
    C. Contact Deposits.
      1. Contact-Metamorphic
      2. Replacement.
    D. Fissure Veins.
  2. Aplite Phase.
    A. Segregations.
      1. Pegmatite.
      2. Tourmaline.
    B. Disseminations.
    C. Fissure Veins.
      1. Quartz-Tourmaline Veins.
      2. Quartz Veins with Sulphides.
      3. Fault Veins.

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[5] Compare Knopf's classification in *Bulletin* 527 *of the U. S. Geological Survey* (1913), which follows:
A. Older Ore Deposits (Late Cretaceous).
  1. Magmatic Deposits.
  2. Contact-Metamorphic Deposits.
  3. Lodes and Veins.
    (*a*) Tourmalinic.
      (1) Silver-Lead.
      (2) Silver-Copper.
      (3) Gold.
    (*b*) Sericitic (May Be Phase of Tourmalinic).
B. Younger Ore Deposits (Not Classified).
Knopf's bulletin will receive further discussion in the descriptions of mines and districts.

3. Quartz-Porphyry Phase.
   A. Disseminations.
   B. Fissure Veins.
   C. Fault Veins.

3. *Rhyolite Period.*
   1. Early Rhyolite Phase.
      A. Disseminations.
      B. Fissure Veins.
   2. Dacite Phase.
      A. Contact Deposits (Redistribution of Earlier Ores).
      B. Fissure Veins.

It will be noted in this that the ore deposition reached a maximum diversity in the granite and aplite stages. In size and widespread distribution, also, the orebodies of these periods are foremost. The types significant of strong igneous activity, the segregations, disseminations, and contact deposits, are most noticeably wanting in the later igneous phases. Deposits traceable to magmatic waters are persistent throughout, and accompany even the later intrusions of small size and slight effect. In general the variety and quantity of ore deposition is directly comparable with the size and importance of the igneous stage within which it occurs.

## PART 2.—THE ORE DEPOSITS

### A. Andesite Period

#### 1. *Andesite Phase*

Orebodies in the andesitic rocks occur as disseminations, a type represented by the native copper of the Baggs Creek district; as contact deposits, illustrated by the Iron Mines of Elkhorn Mountain;[6] and as blanket fissures, either in andesite breccia, as in the case of the Evening Star mine near Elliston, or in amygdaloidal flows as in the Emery (Zosel) district. Many mines throughout the region have developed fissure veins in andesite walls, but in the majority of these, the ore can trace its origin to nearby granite intrusions, the veins of granite origin extending

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[6] The Ward mountain mines, in Madison County, are located upon replacements of schist by quartz, and auriferous pyrite along the flanks of an andesite sill. These mines, of which the McKee and Tucker are the chief, probably fall within the Andesite Phase of the present classification, but the impossibility of determining the age of the sill prevents their inclusion in the above list.

up into the andesite covers. These deposits, therefore, fall under the granite subdivision of the genetic scheme.[7]

1. *Disseminated Deposits.*—Magnetite and native copper are found in disseminations through certain portions of the andesite rocks. The former occurs as a primary mineral, closely associated with the ferro-magnesian silicates, and doubtless represents an early separation of the excess iron from the magma.

Native copper is found in part in similar mineralogical associations, the metal occurring within the augite phenocrysts or groundmass. Frequently, however, it appears in the amygdaloidal upper portions of the flow, with quartz, calcite, and zeolites, and occasionally fills recent cracks in the rock. These latter occurrences cannot be regarded as contemporary with the igneous rock, and it appears probable that wherever found the native copper represents a later concentration of the original copper contents of the flows.

Copper Hill, on Bagg's Creek, east of Deer Lodge, Mont., may be taken as an example of the disseminated copper deposits (see Fig. 3). The metal is found in a rugged hill, which faces the creek in a cliff about 600 ft. high. From the talus slope to the crest, 20 separate flows of andesitic rock can be distinguished, each fine-grained at the base, porphyritic toward the center, and amygdaloidal on top. They dip gently to the northwest, away from the common vent at Thunderbolt Mountain. The rock types are prevailingly basic and represent the earlier stages of the andesite eruptions. Basalt, diabase, hornblende andesite, and augite andesite predominate, the latter rock, rare at the base, increasing in amount toward the hilltop. Andesite breccia to the south and east contains fragments of the basaltic forms, but none of the augite andesite, which, therefore, apparently initiates the later outflows of which the light-gray andesite porphyries and latites are so characteristic. Among the successive flows three are of distinct appearance, with a characteristic pea-green groundmass, and large dark-green augite phenocrysts.

Native copper in appreciable amount is restricted to limited lenses within these flows. The metal occurs in the groundmass, in the augite phenocrysts, and in the amygdaloidal cavities, associated, in the latter

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[7] List of mines in Andesite Phase of mineralization:

Baggs Creek—Copper—East of Deer Lodge.
Iron Mine—Magnetite—North of Elkhorn.
Tacoma—Galena—North of Elkhorn.
Kineo—Galena—South of Elliston.
Hidden Treasure—Gold South of Elliston.
Evening Star—Galena—South of Elliston.
Carbonate Hill—Galena. } Zosel district (Emery)
Butte Caroline—Galena. }
Pierce's Prospect—Galena. }

case, with quartz, calcite, and zeolites (rare). It unquestionably represents a concentration by comparatively cool waters of the copper originally widespread as a constituent (0.02 per cent. or less) of the original rock. The cupriferous lenses, with an average grade of 0.8 per cent. copper, form less than 0.1 per cent. of the mass of the cliff, so that the immediately adjacent rock is entirely adequate to supply the metal required. It is, in fact, probable that this concentration and transportation of the copper took place only within the several flows in which it is now found—a conception supported by the barrenness of adjoining flows of equal porosity, but of slightly different rock type. Joints and cracks, however, have provided channels in which recent surface waters have transported and precipitated the metal. As to the source and character of the original solutions that at once dissolved the copper silicate of the original rock and precipitated locally the native copper of the enriched lenses, little knowledge can be gained. The common explanation of a submarine extrusion, with a lava attacked by sea water, will not suffice for this series of flows upon a land surface. Between the alternatives of a meteoric circulation stimulated by residual heat, and magmatic waters of low temperature, field evidence gives little choice.

2. *Contacts.*—The general contacts of the extrusive andesite with the sedimentary rocks show little metamorphism or mineralization. The intrusive phases, which exert greater influence on the older formations are, moreover, so surrounded by breccias and flows that their contact with the sediments can rarely be seen. Andesite dikes in association with ore deposits are more common, but in most of these instances the age of the andesite is indeterminate.

The best examples of an undoubted contact deposit of the andesite stage are the so-called iron mines on Elkhorn Mountain which have been thus described[8] (Fig. 10).

"On the north side of Elkhorn Peak, there is an iron mine on the contact between andesite and marble, from which a considerable quantity of ore has been shipped to smelters for flux. The ore is along the bottom of the marble mass and is probably a replacement of it. It is a fine-grained magnetite showing grains of chalcopyrite and garnet."

This apparently represents a true contact deposit, formed by the contribution of iron from the igneous rock to the metamorphosed limestone.

3. *Fissures.*—A short distance south of the Baggs Creek deposits of native copper are the mines of the Emery district.[9] These are located upon fissure and blanket veins within the andesite and may be taken as examples of the fissure mineralization of the andesite stage.

[8] R. W. Stone: *Op. cit.*, 97.

[9] See Fig. 3, Map of Emery Mining District.

The ore deposits fall under four heads:

(*a*) Fissure veins in andesite walls
(*b*) Mineralization along bedding of flows.
(*c*) Mineralization of amygdaloidal lavas.
(*d*) Mineralization of andesite breccia.

The three latter types represent an extension of the mineralization from feeding fissures into permeable ground.

(*a*) The main fissure of the district lies in an approximately central position, cutting the flat dipping andesite flows in a northwesterly direction. Its dip is vertical. The ore, about 10 ft. wide, consists of pyrite, galena, sphalerite, and chalcopyrite, in a gangue of quartz and calcite. The walls are tight, with no gouge. Intimate intergrowths of the sulphides, coarse crystallization and occasional quartz-filled vugs, suggest formation in high or moderately high temperatures. Gold and silver are present in appreciable quantities, the latter, within the zone of secondary enrichment, reaching 40 oz. or more to the ton. Development to a depth of 400 ft. has failed to disclose any material change in the vein filling, other than a diminution of the silver as the primary ore was reached.

(*b*) and (*c*) Southward of this main fissure the Carbonate Hill mine has developed a large orebody upon the bedding contact of two andesitic flows. This dips northward at a flat angle (30°). The shoot is about 2000 ft. long and has been followed on the dip for a distance of over 1000 ft.

The vein material varies in width from a few inches to several feet, with tight walls and no gouge. On the foot wall the amygdaloidal andesite has been extensively mineralized; pyritization and silicification of the rock is thorough; and the amygdules contain a filling of quartz, calcite, galena, spalerite, millerite, arsenopyrite and some zeolites. Veinlets of quartz and the sulphides are also common throughout this altered rock. The vein itself is well banded; layers of the quartz, calcite, and rhodochrosite gangue alternate with bands of the sulphide minerals, with frequent vuggy faces of quartz and calcite crystals. Apparently at least one reopening of the fissures is represented. The sulphide minerals, pyrite, galena, sphalerite, chalcopyrite, millerite and arsenopyrite, form intimate aggregates of much finer grain than the typical ore of the vertical fissure, which, together with the occasional presence of zeolites, indicates lower temperatures during the formation of the bedded deposit. The hanging-wall flow, a dark hornblende andesite, is entirely unaltered.

Economically this orebody has been exploited for the silver which in the oxidized zone and for several hundred feet below water level has been enriched to a high grade. On the lower levels the diminution in silver was accompanied by an increase in sphalerite, and the ore shoot, while persistent geologically, ceased to be of value to the miner.

(*d*) A small mine north and east of the Carbonate Hill, in the same

district, affords a convenient example of mineralization in the andesite breccia, although larger deposits are known elsewhere.[10]

In the type example, a layer of andesite breccia of irregular width is intercalated between a light-gray andesite flow and a dark hornblende andesite. The whole series dips 25° to the northeast, and movement has taken place along the top of the breccia. The broken and porous zone thus created has been penetrated by mineralizing solutions, which have selectively attacked the breccia groundmass and fault plane. Excess of galena and extreme fineness of texture mark this deposit and testify to the cool conditions of its formation. It carries much more gold than the blanket or fissure-vein deposits.

4. *Summary.*—In general, therefore, the andesite stage of igneous activity concludes with two types of mineralization; the first, a zeolitization of the igneous rock and concentration of its copper content, probably effected by the magmatic waters of dying vulcanism; the second, a contribution by magmatic solutions, which deposited in such favorable localities as fissures, bedding planes, amygdules, and breccias, a representative sulphide ore carrying universally galena, sphalerite, pyrite, chalcopyrite, arsenopyrite and millerite with appreciable amounts of the precious metals.[11] The distinction between the deposits formed by concentration of metal originally disseminated throughout the rock and those derived from the metalliferous residue of the igneous reservoirs is a marked one, both in mode of occurrence and in character of mineralization.

## B. Granite Period

### 1. *Granite Phase*

1. *General Description.*—The earliest ore deposits of the granite phase are the segregations from the molten magma. These may be basic, as magnetite and auriferous pyrrhotite at the margins, or acid, as the quartz segregations within the batholith. Disseminations, mainly of auriferous or cupriferous pyrite, near the basic borders of the granite, are essentially contemporaneous with these. Contact deposits, representing a contribution of metalliferous waters from the intrusive rock, follow closely in point of time. They may be subdivided genetically into the high-temperature deposits marked by garnet, tremolite, and the metamorphic minerals, with associated copper mineralization, and the lower-temperature deposits, generally of auriferous pyrite, replacing the intruded rocks to some distance from the granite contact. Fissure veins,

[10] The Evening Star mine south of Elliston.

[11] The gold-pyrite veins of the Radersburg district (Fig. 4) are not included here because although entirely within the andesite walls they show strong affinities with the granite type of vein filling and a source of the solutions within the granite is rendered possible by the proximity of such intrusions.

formed subsequently to the crystallization of the granite, mark the last stages of its mineralizing activity. They are confined to the uppermost portion of the granite intrusions, and generally extend well into the overlying rocks. Quartz and pyrite, carrying gold, are the common minerals.

2. *Segregations.*—The approach of the basic rim of the granite to the extreme forms of pyroxenite, peridotite, and even pyrrhotite is not rare. It apparently occurs most frequently where a forerunner of the magma has widened into a peninsula, nearly surrounded by cool sedimentary rock.[12] Under these conditions the borders of the igneous peninsula display the most basic phases, and in many cases pass into pyrrhotite at the contact itself. The ultra-basic types, and especially the pyrrhotite, may carry gold to a commercial degree.

In the Spring Hill mine near Helena, the characteristics of such a deposit are well shown.[13] An outlying granite boss penetrates massive Carboniferous limestone with many irregular contacts, at least one projecting mass of the intrusive rock being well surrounded by the sediments. The igneous buttress shows basic faces on two sides with a central core of quartz-diorite. The gradation in either direction from the normal rock through diorite, peridotite, pyroxenite and pyrrhotite is a gradual one, covering 50 to 75 ft., and the gold content increases with equal uniformity as the contact is approached. The limestone is metamorphosed and cut by large and small dikelets of the basic rock, but the ore deposit is restricted to the igneous rock itself.

A similar pyrrhotite orebody occurs in the Golden Curry mine near Elkhorn, and is well described by Knopf.[14] In this mine magnetite, as veins and segregations within the granite near its contact faces, affords the single somewhat doubtful example of this mineral as a product of igneous differentiation.

Contrasted with these ultra-basic separations from the magma are quartz segregations found with frequency throughout the granite areas. The largest, several hundred feet in diameter, lies a mile east of the town of Basin.[15] It consists of massive vitreous quartz, with minute amounts of tourmaline and molybdenite, surrounded by typical unaltered coarse quartz-monzonite. Below the quartz mass is a porphyritic rock composed largely of orthoclase and quartz, probably representing the aplite stage. This has been penetrated by fingers of the quartz rock, which locally approach the composition of pegmatite.[16] It is, therefore, probable that

[12] An interpretation suggested by F. A. Linforth in oral communication.

[13] Ref. report by F. A. Linforth, A. C. M. Geological Department, Butte, Mont.

[14] A. Knopf: Ore Deposits of the Helena Mining Region, Montana, *U. S. Geological Survey Bulletin* 527 (1913), 137–139.

[15] Many similar but smaller quartz masses are found between Boulder and Elkhorn. Others are known north and west of Butte.

[16] See also A. Knopf: *Op. cit.*, 123.

this acidic mass is much more recent than the basic contact differentiations, and represents an injection of siliceous residue rather than magmatic segregation. The molybdenite and tourmaline, however, imply true igneous or pegmatitic conditions of origin.

3. *Disseminations.*—Disseminated ores in the granite consist almost entirely of auriferous or cupriferous pyrite. While largely the product of the primary crystallization, these deposits frequently show, by silicification and sericitization of the rock, that hydrothermal waters have contributed to their extension and enrichment. They therefore fall naturally, in chronological sequence, between the magmatic segregations and the later deposits of true hydrothermal origin. The more important occurrences of such ores are found in the Cardwell mining district, northeast of Whitehall; in the Heddleston mining district north of Helena, and on Red Rock Creek, near Basin.

In the Red Rock Creek occurrence the pyrite is entirely a primary mineral[17]—as thin sections show—in a fine-grained basic phase of the granite, near its contact with the overlying andesite. The pyrite is of small size, and not abundant, but is of widespread distribution. It carries enough gold to bring the grade of the general rock type locally up to about 50 or 75 c. per ton. The Heddleston district shows similar basic granite, with cupriferous pyrite to the amount of 5 or 10 per cent. of the rock by weight. The gold content is but a few cents per ton, but secondary enrichment makes small commercial orebodies of disseminated chalcocite.

The ore of the Golden Sunlight mine near Whitehall owes its value to a hydrothermal enrichment[18] of a similar pyritic phase of a laccolith, intruded into and between quartzitic shales. Its upper surface is composed of fine-grained granite porphyry, almost devoid of quartz, and abundantly speckled with primary pyrite. This rock and the overlying sediments are cut by numerous tiny fissures, which have served to introduce the later solutions. The regions traversed are thoroughly sericitized and silicified; veinlets of quartz and pyrite abound; and the brecciated or otherwise porous portions of the slates are thoroughly impregnated with the quartz-pyrite mineralization. In this way the gold originally disseminated throughout the zone of the primary pyrite has become concentrated along the subsequent lines of enrichment and dispersal.

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[17] The rock in which this disseminated pyrite occurs is a basic quartz-monzonite. The minerals, in a holocrystalline aggregate, are plagioclase, orthoclase, hornblende, biotite, magnetite, and quartz. Magnetite, the earliest mineral, is unusually abundant, while quartz is rare. The rock is entirely unaltered, showing neither sericite, chlorite, or kaolin. The rare pyrite is probably an original, pyrogenetic mineral, occurring in irregular aggregates very similar to the magnetite masses.

[18] In the Moffett Mine, in Bear Gulch, a similar hydrothermal enrichment is indicated by the presence of tiny seams of chalcopyrite in the basic rim of the granite (F. A. Linforth in oral communication).

The fundamental factor controlling the formation of the disseminated ores is therefore much like that observed with the segregations, the presence of a well-developed basic phase of the granite. In the case of the segregations a peninsular mass thrust out into the intruded rocks is most favorable to the differentiation of an ultra-basic contact; with the disseminated ores the common association is with the crust of the granite, preferably in outlying areas of the intrusive.

4. *Contact Deposits.*—The orebodies that owe their existence to contact action of the granitic magma may be divided into high-temperature and low-temperature contact replacement deposits. The former include the ores closely associated with the high-temperature contact minerals and are generally at or near the contact itself; the latter represent the orebodies formed at greater distances from the intrusive rock by the circulation of solutions of magmatic source.

In the Georgetown district (Fig. 6) west of Anaconda, both types are represented, and their inter-relation is well shown. Of the first group are the orebodies of the Cable mine[19] contained within an inlying mass of metamorphosed limestone. The sedimentary rock has been highly altered, the contact phases ranging from nearly pure, coarsely crystalline calcite to aggregates of garnet, actinolite, magnetite, pyroxene, specularite, epidote, and mica. Large areas of magnetite alone are also frequent. Into this metamorphosed rock magmatic contributions of silica, iron, copper, bismuth and gold have been introduced in the form of quartz, pyrite, chalcopyrite, tetradymite, and metallic gold. The ore, therefore, consists of those minerals, together with calcite and small amounts of the contact silicates, irregularly distributed throughout the metamorphosed area. Their location in detail was determined by the presence of fissures, but the orebodies are in part at least the result of direct contact metamorphic action. Gold is in general the valuable constituent, but some copper ore has been shipped. Most of the many similar deposits on the borders of the granite, however, have been exploited for the secondary enrichment of chalcopyrite, gold being of subordinate importance. The Highlands, French Gulch, Tidal Wave and Ophir districts contain many good examples of this type of ore formation.

A short distance from the Cable mine are the deposits of the Southern Cross[20] district. They occur within a crystalline phase of magnesian limestone somewhat farther from the granite contact than the Cable ores. The high-temperature silicates are absent, and the limestone has received no direct accession of silica or iron from the igneous magma,

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[19] W. H. Emmons and F. C. Calkins: Geology and Ore Deposits of the Philipsburg Quadrangle, Montana, *U. S. Geological Survey, Professional Paper* 78 (1913), 222.

[20] Emmons and Calkins: *Op. cit.*, 231.

Paul Billingsley: Southern Cross Mine, Georgetown, Mont., *Trans.* (1914), **46**, 128.

whose effects are limited to a thermal recrystallization of portions of the sediments. Small dikes and sills are rather numerous, but apparently the metamorphism and ore deposition depend upon the main mass rather than these minor offshoots. The limestone is traversed by numerous fissures of slight displacement which have served to introduce the mineralizing solutions from the granite into the permeable crystalline limestone beds. As originally formed, the orebodies consisted of a pyritic replacement of this rock along the intersection of favorable beds with the fissures. The mineralization was pyrite, magnetite (?), siderite, calcite and quartz, with gold and copper present in small amounts. The orebodies have been peculiarly susceptible to the effects of oxidation, and in addition to the reconcentration of gold in the original shoots many mechanically formed deposits of auriferous cave breccia and corroded limestone have resulted.

In general, therefore, the Southern Cross district represents a more remote phase of contact action than the Cable. The contact silicates, with their addition of iron, alumina, and silica from the magma, are absent, and the limestone shows only the slight recrystallization caused by the heat of the igneous intrusion. The ore minerals, instead of being partly contemporaneous with the contact phenomena, are of subsequent infusion, and owe their source to residual granitic solutions rather than to the high-temperature emanations of the contact itself. In short, the Southern Cross type lies midway between the pyritic replacements of contact metamorphism and the siliceous fissure fillings of the cooling granite.[21]

In the Gold Coin mine, the connection with these latter veins is well-nigh completed. Ores of the Southern Cross type are here found in a fissure in unaltered limestone. There is no replacement of this rock, and the deposit approximates in form and mineralization the true quartz-pyrite gold veins of the granite stage.[22]

5. *Fissure Veins.*—The last ore-forming activity of the granite period is represented by fissure veins, largely in the adjoining rocks but in many cases penetrating the granite itself for a few hundred feet from the contact. These veins are filled primarily with quartz and pyrite, but calcite, chalcopyrite, sphalerite, and galena, are occasionally found. The type is widespread, and carries the ores of many districts, among which Garnet (Fig. 7), Scratchgravel (Fig. 8), Marysville (Fig. 9), Unionville, Hassel, Pony, Bannack, and Whitehall are the more important.

The mines of the Garnet district surround an outlying "cupola" of granite, intrusive into Cambrian limestone and quartzite (Fig. 7). The

[21] The Red Lion district mines (Fig. 5) are entirely of the Southern Cross type of ore deposit.

[22] A list of districts in which ore deposits of each type are found is appended to this paper.

veins occur within a limited zone including the peripheral portion of the granite and the neighboring sedimentary rocks (Fig. 21). Five hundred feet within the granite, and possibly twice this figure beyond its borders, will cover this area of mineralization. The strike of the fissures, which are rather narrow (less than 2 ft.), is approximately parallel to the borde of the granite, and the flat dips toward the igneous rock are nearly perpendicular to its contact. Despite the varying wall rocks—granite, limestone, quartzite, and andesite (sills in the quartzite)—the mineralization is uniform throughout the district, and minerals other than the prevailing quartz and auriferous pyrite are rare. Primary ore shoots, marked by an increased width and high percentage of pyrite, are numerous and persistent in depth within the limits of the productive zone. Between and beyond these the fissures are represented by only the most minute cracks. Enrichment of the orebodies by descending solutions is universal and is farthest advanced at the bottom of the oxidized zone, and in the partially attacked ore immediately below. At the surface itself the gold is absent, and it is probable that the enrichment has been accomplished mechanically by groundwater circulation. Galena and chalcopyrite have been recorded from the bottom of the veins developed deepest within the granite. In general, therefore, the Garnet veins consist of fissures formed in the solidified crust of a granite boss and in the adjoining intruded rocks, filled by the products of solutions derived from the more deep-seated portions of the igneous rock. The fissure zone is distinctly a halo of the periphery of the granite mass, and the uniform vein filling in diverse wall rocks implies contribution from a common source.

The Marysville district likewise surrounds a small outlier of the main batholith (Fig. 9).[23] The intruded rocks are calcareous and argillaceous slates of Algonkian age. Less extensive erosion than in the case of Garnet has left much of the roof in place, and the details of contact are more completely disclosed in mine workings. Here also the occurrence of mineralized veins coincides with the periphery of the granite and the adjoining metamorphosed sediments, the ores ranging from about 500 ft. within the granite to about 1500 ft. beyond the contact (Fig. 21). The veins occur in two systems which in a general way are respectively radial and tangential to the igneous contact. Their dips are steep, as is the contact also. Later faults are sometimes slightly mineralized.

The vein filling in Marysville is essentially quartz, with subordinate amounts of calcite, fluorite, galena, pyrite, sphalerite, and chalcopyrite. The quartz, often either chalcedonic or amethystine, has in the typical deposits replaced an earlier stage of lamellar calcite. Open cavities and

[23] For detailed descriptions of many of the more important mines of the district, for longitudinal and cross-sections of veins showing their relation to the granite, and for history of the district, see paper by C. W. Goodale, The Drumlummon Mine, *Trans.* (1914), **49**, 258–283.

comb structure, with crystals of quartz of later generations, are abundant. Fragments of country rock found in many of the veins suggest that the fissures, prior to the mineralization, existed as cavities filled with detritus. On the lower levels of the Drumlummon mine, the vein consists entirely of such a detrital "breccia," cemented by lamellar quartz and calcite.[24] Post-mineral faulting along the vein courses is common. The sequence may be summarized as follows:

1. Opening of fissures and partial filling by detritus.
2. Impregnation of fissures by lamellar calcite.
3. Replacement of calcite by quartz and sulphides.
4. Early faulting, partly mineralized.
5. Later faulting, unmineralized.

It follows that the quartz mineralization of the present orebodies is considerably more recent than the original formation of the fissures, which, it can scarcely be disputed, closely succeeded the intrusion and cooling of the granite mass. The characteristics of the Marysville veins, moreover, are those of deposits formed by relatively cool waters at shallow depths. The open cavities, chalcedonic quartz and absence of hot temperature sulphide minerals except in small quantities, are significant in this connection. The 1600 ft. of depth reached by the Drumlummon mine witnessed the disappearance of the typical primary mineralization of the upper levels, and quartz, calcite, chalcopyrite, and tetrahedrite—the sulphides in very small amounts—alone persisted (see Fig. 18).

In seeking a source for this post-granite, shallow depth mineralization, two possibilities stand out. One of the more recent igneous periods may have contributed these solutions to the fissures of an older period or the granite itself may have yielded them from internal sources. The distribution of the Tertiary igneous rocks renders it impossible to trace the many instances of the Marysville type of vein filling to such a source. This would require that the solutions travel for long distances, in a region where cavities and fissures abound, to group themselves in a series of veins surrounding a granite intrusion. On the other hand, the granite and its associated differentiations afford a source available to every vein in the district. The mineral association is similar to that found in recent and active hot springs of granitic origin. Even the replacement of calcite by quartz is not without parallel among the gold quartz veins of the granite batholiths of the Cordilleras.[25] It seems probable, on the whole, therefore, that the Marysville veins represent a late contribution from the granite, wholly analogous to hot springs now found around the batholith, and that while possibly they are as young or younger than the more

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[24] A "breccia" of this type does not necessarily involve displacement of the fissure walls. It exists in Butte, Georgetown, and Radersburg under circumstances that place its origin in the filling by wash or attrition of open cavities.

[25] DeLamar Mine, Idaho. See Lindgren's *Mineral Deposits*, 437.

recent igneous stages, they must still be genetically classified with the granite period.[26]

The Garnet district, with its deposits of a type characteristic of formation at fairly great depths, lies at an elevation of about 6000 ft. The surface at the time of vein formation must have exceeded 7000 ft., the level of the subsequent Eocene peneplain in that region. At Marysville the peneplain lies at 7000 ft., while the veins outcrop at 5500 to 6000 ft. elevation. The post-Eocene erosion is thus essentially the same in both cases. The location of the veins at both Garnet and Marysville is similar in reference to the igneous rock. The great differences in vein type are thus again carried back to a difference in time, with the Marysville mineralization deposited at slight depths probably after much of the post-Eocene erosion had occurred.

6. *Summary.*—The ore deposits of the granite period are associated with the outlying "cupolas" and bosses rather than with the main mass, forming on the map a halo around the larger areas (see Fig. 1, 23). The true significance of this lies in the relation of the smaller intrusions to the batholith. These are most abundant on the flanks of the eroded mass, diminishing in number and size with distance from the contact. There is evidently a gradation downward from small dikes, the vanguards of the intrusion, through a phase of larger irregular stocks and bosses to the connected mass of the batholith. The vertical distance from the remotest advance intrusions to the main mass is in general less than 2000 ft. Within this zone the phenomena of the granite mineralizations have occurred; segregations on the borders of the protruding tongues of magma; disseminations in basic border phases; contact-metamorphism accentuated by contributions of iron and silica from the magma; replacements of the intruded rocks by the iron-silica solutions; and fissure filling by the same products of magmatic segregation, all

[26] This theory has already been advocated for the views of the Marysville district by Barrell and Weed, and is well summarized by the latter. "There are many productive mines working veins cutting the igneous rock and the contact rocks above them. Such vein fissures are caused both by the contraction due to the crystallization and cooling of the igneous rock and by the shrinkage of the metamorphic zone above the igneous rock. As the magma and its surrounding shell of heated sediments cools down it must contract, and this contraction will result in a cracking both of the igneous rock and the contact zone; and if the rocks of the contact zone are homogeneous the cracks will assume a more or less radial position. If these cracks extend to a depth sufficient to reach this molten magma, they will be filled and dikes will be formed; if not, cracks become channels for pneumatolytic vapors and later circulating waters, and thus pegmatitic veins and true mineral veins may be formed and may merge into one another."

This hypothesis is disputed by Knopf in the case of Marysville on what seem to us insufficient grounds, and upon due consideration and investigation of the many districts similarly situated we feel that Weed's and Barrell's stand is strengthened rather than weakened by the additional evidence gathered.

are in the main limited to the smaller masses. It is evident that in these, magmatic instability, witnessed both by basic segregations and acid residues, was accentuated to a degree never reached by the more slowly cooling mass underlying.

## 2. *Aplite Phase*

1. *General Description.*—Aplite appears within the batholith in three phases: (1) The earliest are segregations on the contact, and acidic dikes in advance of the main intrusion. (2) Following these in time come the main aplite injections—bodies of small dimension and irregular outline, most abundant in the peripheral portions of the granite, and forming in general about 10 per cent. of the total mass. The contact indicates that the granite was still hot when these intrusions occurred, and the absence of connecting links suggests that the aplitic material was a residue of widespread local occurrence in the crystallizing batholith. (3) Later dikes of more basic rock, quartz-porphyries or pyroxene aplites, are believed in some instances to be offshoots of large aplitic masses below. It is believed that these larger deep-seated bodies are of identical origin with the smaller injections of the outer crust, but that their size and position resulted in slower cooling and partial differentiation, of which the porphyry dikes are the product. These show by their contacts that the granite was cool at the time of their intrusion. The importance of the associated mineralization has led to the creation of an independent group in the genetic classification for such quartz-porphyry.

2. *Segregations.*—A siliceous residue itself, the aplite is particularly susceptible to a differentiation and segregation of its acid elements. In the outlying dikes it is not rare for the rock to grade from the aplite into alaskite and even unalloyed quartz at the extremities.[27] Throughout the aplite areas segregations of pegmatite, often highly siliceous, abound, and igneous quartz, with traces of molybdenite and tourmaline, is not unknown. The smaller dikes within the granite pass into quartz or tourmaline or both with no crystalline break. Sulphide minerals are occasionally intergrown with the tourmaline.

These conditions can result only from a close association of high-temperature solutions and vapors, rich in silica and boron, with the aplitic magma. They must, in fact, be regarded as contemporaneous emanations from it. In part, these solutions are caught and crystallized within the cooling rock itself; in part, they escape and, being more mobile than the aplite, penetrate the surrounding rock in the form of pegmatite and quartz-tourmaline dikes.

In the Valley Forge mine, at Rimini, and in general throughout that district as well as in the Oro Fino district near Deer Lodge, some sulphide

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[27] Lost Creek, Deer Lodge, Mont.

minerals are found intergrown with the quartz-tourmaline segregations from aplite dikes.[28] Galena, sphalerite, pyrite, and chalcopyrite are known in this association. Such ores are of little economic importance, however, the vast bulk of the sulphides occurring with a later quartz mineralization, which also carries some tourmaline.[29] The earlier igneous ores with their parent aplite may be seen, in the Valley Forge mine, as a

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[28] Gold is the only mineral of economic importance found and is associated with the pyrite, which is the most abundant sulphide. Abortive attempts have been made in the Oro Fino district to stope gold ore from these quartz-tourmaline-pyrite dike veins. Possibly the unusually high gold content of the lead-zinc-silver ores of the Rimini district is also connected in some way with the tourmaline dike type of mineralization.

[29] This differentiation between the tourmaline-quartz-pegmatite dikes and the sulphide veins which also show some tourmaline, is opposed to the conclusions reached by A. Knopf, in *Bulletin* 527, of the U. S. Geological Survey. It is adhered to by the authors of this paper, however, for the following reasons:

I. Throughout the Rimini district there are many evidences of the priority in age of the tourmaline to the lodes. For example: (*A*) In the Valley Forge mine the tourmaline occurs in a zone of many parallel bands with strikes nearly east-west, while the vein traverses the belt with a course of N 70° E, cutting the successive tourmaline dikes at an acute angle. In detail the vein can be seen at many points truncating tourmaline bands. The vein proper is formed by mineralization along a fault fissure, which is definitely subsequent to the tourmaline period. (*B*) In the Merritt mine a drag block of tourmaline, with some quartz and sulphides, lies between two fault slips, which carry the later sulphide mineralization along them. (*C*) In almost all the mines a breccia of aplite, quartz and tourmaline, cemented by an ore of quartz, tourmaline, and sulphides is of common occurrence.

II. Two types of tourmaline are present. The earlier (as is evidenced by structural contacts) is coarse-grained and of uniform texture across maximum widths of 10 ft. Its contacts with the granite are sharp and definite. The later is present as a component of the "black quartz" that accompanies the veins. It is of microscopic size and forms an irregular proportion of the aggregate.

III. There is much evidence of the contemporaneous formation of the tourmaline proper and the aplite (*A*) .400 ft. south of the Lee Mountain vein, are unaltered aplite dikes in fresh granite. They show numerous crows-foot segregations of tourmaline. Near the Comet mine also such segregations in unmineralized aplite are common. (*B*) Dikes of aplite occasionally grade with crystalline continuity into pegmatite, quartz-tourmaline, and coarse-grained tourmaline.

IV. There is no uniformity of distribution between the tourmaline dikes and the veins. (*A*) On the Sixty-ninth Hill tourmaline is found at points remote from the Lee Mountain vein on either wall, and with varied strikes. In many of these cases no associated mineralization whatever can be observed. (*B*) In the Stanton mine very rich lead-zinc-silver ore is found, sometimes up to 5 ft. wide with no tourmaline either associated with the ore or present in the vicinity. (*C*) In the John McGrew prospect the "black-quartz" type of tourmaline occurs along a fault vein, with none of the coarse-grained tourmaline near. In brief, the quartz-sulphide ores are found throughout the district. The "black quartz" sometimes accompanies these ores, but not invariably. The coarse tourmaline occurs almost exclusively in the northern part of the district, and is here seen to be earlier than the vein mineralization where the two happen to intersect.

fragmental breccia cemented by the later hydrothermal minerals. In general, therefore, the metalliferous tourmaline segregations are merely significant antecedents of the great tourmalinic ore-bearing mineralization that followed.

The single instance of economic exploitation of a quartz segregation is the Quartz Mass, previously described. This, said to run 99 per cent. silica, was formerly used as converter lining in the Butte copper smelters.

3. *Disseminations.*—Pyrite and chalcopyrite are common as primary minerals of the aplite, and molybdenite is a frequent constituent of the pegmatitic phases. In general, however, these sulphides are in very sparse quantity.

The Modoc mine, in the Red Lion district, affords the single instance where a disseminated sulphide has raised the aplite to a value sufficient to encourage prospecting (Fig. 5). This mine has developed an aplite boss of elliptical plan, with axes of 100 and 50 ft. respectively, which is intruded along the contact of a granitic batholith. The boss contains much primary pyrite and chalcopyrite in a rock of typical aplite constitution. On the surface, and at shallow depths, oxidation and enrichment are apparent, with the chalcopyrite showing alteration halos of bornite and chalcocite. On the lowest level (200), however, the rock is unaltered. Within the zone of secondary enrichment the boss carries about 1.5 per cent. copper but the primary grade is less than 1 per cent.

4. *Fissure Veins.* (*a*) Origin.—The distribution of the aplite intrusions throughout the batholith is also that of the great quartz mineralization that followed. In the majority of instances the vein-forming solutions have penetrated the same fissures that the aplite had earlier found, or have occupied fractures in the aplite, reopening along the same general zones. The aplite, in its ultra-siliceous phases, forecasts, with its quartz, tourmaline, and sulphide segregations, the composition of the later ores. In more than one case the boundaries of an aplite dike on the surface become the boundaries of a later quartz vein below, the latter feathering out into quartz stringers in the dike at intermediate points. Identity of position, similarity of composition, and deep-seated origin, therefore, imply a common source for the aplite magmas, the pegmatitic vapors and the vein-forming siliceous waters. Their chronological sequence suggests that the several stages represent increasingly acid residues from the local reservoirs of aplite magma.

(*b*) Distribution.—The occurrence of these veins of aplite source is widespread. They contain most of the ores of the Elliston, Rimini (Fig. 13), Lump Gulch, Wickes, Comet, Basin, Jack Mountain (Fig. 12), Little Boulder and Homestake districts, and in part those of the Butte district. They are found several hundred feet above the granite contact, and their barren roots are exposed in the deepest portions of the batholith laid open by erosion. Nine-tenths of the silver, lead and zinc ore of the

region exclusive of the Butte district has been extracted from veins of this series.

(*c*) Character.—The following characteristics are universal:

1. Position near the contact of the main batholith, with fissures mainly within the granite.
2. Strike approximately east and west.
3. Dip steep.
4. Many parallel and reticulated fissures rather than a single large opening.
5. Extensive replacement of the wall rock by the mineralizing solutions.
6. Sericitic alteration of the wall rock.
7. General presence of the sulphides of all the base metals—lead, zinc, copper, and iron, in all the veins.
8. Predominance of each metal in turn in an orderly succession proceeding from the source of the solutions.
9. Increase in proportion of quartz relative to total sulphides with increased depth.

A fuller description of these common traits may be of profit.

1. The great majority of the ore deposits contained in these veins of aplitic origin are situated within a short distance, vertically, from the contact of the main granite mass. Some, such as the Alta and Gregory, extend upward into the intruded rock, but the majority, among which can be numbered the Bertha, Comet, Eva May, Crystal Bullion, Hope, Valley Forge, Lee Mountain, Ontario, and Monarch, are within the granite at distances less than 1000 ft. from its upper surface. The few veins more deeply placed contain little but quartz and suggest the eroded roots of veins of normal type above. One hundred and nine veins outside of the Butte district have been identified as belonging to this group, and wherever possible the position of their outcrop with reference to the granite contact has been determined. The restoration of the contact has been accomplished in many cases by connecting residual fragments of cover; in other instances, as at Marysville and Philipsburg, the physiographic evidence of drainage has been used; in still others the limit of metamorphism in the sedimentary rock is significant; but in a surprisingly large number of places the contact itself is preserved and gives a definite plane for measurement. In the Wickes, Comet, Basin, Jack Mountain and Rimini districts this is largely the case. The results of the classification of these orebodies according to position are striking. Of the 109 listed, nine veins could not be definitely placed with reference to the top of the granite. Of the remaining 100, 15 are contained for the most part in the overlying cover, 54 lie within a distance of 500 ft. below the contact, 28 outcrop between 500 and 1000 ft., and including the deep developments on the above, only four veins are known at depths below

1000 ft. With the exception of the Granite-Bimetallic vein, replacement ores in limestone, and possibly some of the Butte silver veins, the productive portions of these fissures are limited to the region between 1000 ft. above and 1000 ft. below the contact (Fig. 22 and 24).

2 and 3. The close approach of the strikes of these veins to an east-west course, and the prevailing steep dips, cannot be without significance. A reconstruction of the granite surface indicates that above the vast majority of these veins the contact lies at a flat angle, to which the steep dips are approximately perpendicular (Fig. 2 and 17). The main granite mass, within which the greater part of the veins lie, is shown by such a reconstruction of its original form to be elongated on its summit, in a north-south direction, while comparatively narrow between its eastern and western contacts. These are appreciably steeper than the northern and southern boundaries.[30] The fissures, therefore, are normal in strike to the long axis of the intrusion, and lie at high angles to the two roughly parallel bounding contacts on the east and west. Their distribution, coextensive with the granite, suggests a common origin connected with the dynamics of that rock. These several requirements are fulfilled by regarding the fissures as contraction cracks formed in the outer mile of the cooling granite, following in their position the habit of such fractures.

4. The detailed structure of the veins further indicates that no great displacement accompanied their formation.

The structure of several typical veins of this series is shown in Fig. 14, 15 and 16. By the use of solid lines for vein material, quartz, sulphides, etc., and dotted lines for post-mineral faulting, it is attempted to represent the character of the fissure, the distribution of its mineral contents and its influence on subsequent movement of the ground.

The Crystal vein is at once the simplest and most representative of the type (Fig. 14). Its total developed length is about 6 miles; its average width about 30 ft., and it has been explored from within the andesite roof blocks to a depth of about 1500 ft. within the granite. The section shown in the Crystal mine is typical of the entire extent. Here the total width of mineralization is about 50 ft., with a bordering zone on either side of silicified and sericitized aplite. Within this 50 ft., bands of true fissure filling—quartz and sulphides—alternate with belts of mineral-

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[30] While in a general way the top contact of the granite is nearly flat, sloping gradually under the cover rocks in quaquaversal manner, nevertheless the dome has a distinct long axis, with a general north-south direction. Along this axis the top is nearly level, with local irregularities, and when the granite finally goes under cover to the north and south its shallow depth is suggested by the numerous and comparatively large outliers that mark its course (see Fig. 1). On east and west, however, such outliers are both fewer and smaller, indicating a deeper submergence of the granite on these flanks, and hence a steeper dip to these lateral contacts. A contour map of the restored granite surface shows graphically the longer north-south axis, and the steeper dips on the northwest and southeast contacts of the main intrusive mass.

ized country rock traversed by numerous tiny stringers. The vein bands average 2 to 5 ft. wide; the intervals may run up to 10 ft. Along the course of the vein great variation is found. The several quartz-sulphide fissures terminate or finger out, and new bodies appear in the mineralized aplite between. The richer ore ceases in one belt, to begin in another hitherto barren. In short, while the zone itself is continuous, the separate fissures and ore shoots that compose it are of limited extent, existing only as overlapping lenses and reticulated cracks within a belt of silicified and sericitized country rock. The belt has in the Crystal, as almost invariably throughout the region, been followed by post-mineral faulting, which closely parallels the banding of the vein system.

In the Comet mine the general zone of mineralization is unusually wide—nearly 150 ft.—and may be subdivided into three minor belts (Fig. 15). Each of these has the same characteristic features that are found in the Crystal. On the lower levels they diverge, forming three parallel mineral zones rather than a single wide one. The rock alteration in the Comet is more pronounced and extensive than in the Crystal, a fact in accord with the obviously greater fissuring and more intense mineralization at the former mine. The Comet vein is known for a distance of about 5 miles, with a width of 100 ft. and a vertical development from the andesite contact to a depth of about 1000 ft. within the granite. The Baltimore vein illustrates a feature also found in Butte—the spraying, as it were, of the solutions from the main fissure into a series of smaller cracks at large angles with its course (Fig. 16). This structure, called "Horsetail" in the copper mines, implies the presence of permeable fissures which were traversed by the major vein openings, and filled by the mineral solutions. The close association of these zones with the larger east-west veins, and their detailed structure in the Butte district, indicates that their origin is in some way traceable to the dynamics of the major fissuring, of which they may be mechanical resultants. The "Horsetail" structure is at any rate more than a local feature, and must be due to some widespread and fundamental cause.

In general, then, the veins of the aplite stage fill the openings along wide zones, often several miles in length, and composed of many small overlapping fissures (see Fig. 12). The present continuity and unity of the veins is due to the diffusion of the mineralization along these indefinite zones, the cementation with quartz and sulphides of disconnected fissures, and the silicification of the intervening permeable country rock. Post-mineral faulting, following the lines of weakness these zones provide, gives a false appearance of definite and extensive walls to the orebodies.

The fissure zones which have thus determined the position and character of these ore deposits of the aplite state apparently represent a separation of the walls along their course. This is suggested by the

short lenticular openings as well as by the absence of evidence of movement, such as shearing in the intervening country rock, or displacement of intersected dikes of aplite. Such a drawing apart must imply a state of tension in the granite crust, a state which of course existed during its cooling and contraction. So from this angle also the origin of these fissures seems to fall in with the conclusions already arrived at—that they mark the lines where the shrinkage of the batholithic crust found expression in tension cracks.[31]

5 and 6. It has already been observed that there is intense alteration of the country rock in the general vein zone. The new minerals formed are sericite, pyrite, and secondary quartz; the old minerals attacked are the feldspars and ferromagnesians. Occasionally needles of tourmaline are added. Knopf finds in the Rimini district an actual gain in silica, iron, and sulphur, and a loss in all the bases except potash.[32] The actual gain in mass by his analyses is 8.87 per cent. of the original. The altered rocks, of which this may be considered representative, are thus marked by the absence of any dark minerals (unless tourmaline is present), by a high proportion of quartz, and by numerous specks of pyrite. The limits of alteration are indefinite, the change penetrating along joints and fissures far beyond the main belt. In general, the size and intensity of mineralization of the ore deposit is directly proportional to the amount and extent of sericitic rock alteration. Many of the smaller veins and shoots show only a few feet of pyritized granite along their walls. Chloritic alteration is common along the post-mineral faults.

7. The primary vein filling of the aplite stage includes the following minerals: quartz, rhodochrosite, rhodonite, tourmaline, calcite, siderite, and fluorite, in the gangue, and pyrite, arsenopyrite, galena, sphalerite, chalcopyrite, tetrahedrite, bornite, stibnite and argentite, among the sulphides. Of known occurrence, but rarely encountered, are hübnerite, molybdenite (oftenest with pegmatite), proustite, pyrargyrite, gold, magnetite, hematite, and cassiterite. The great majority of the ores consist of the sulphide minerals listed above in varying proportions, in a gangue of quartz.

Using the same examples previously considered with regard to position of outcrop, it appears that the mineral distribution is as follows (detailed information is available in 93 cases):

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[31] Prof. Clapp, of the Montana State School of Mines, argues (in oral communication) that the state of tension was due rather to great dynamic stresses, probably those which found expression later in the Basin Range normal faulting. The suggestion is doubtless of value, and may be an additional explanation of the fissure occurrence. It has seemed to the authors, however, that the interpretation given in the text is at once simpler and sufficient.

[32] A. Knopf: Ore deposits of the Helena Mining Region, Montana, *U. S. Geological Survey, Bulletin* 527 (1913) 50.

93 *Fissure Veins of Aplite Stage*

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|---|---|
| Pyrite | is present in 86 veins, or 92 per cent. |
| Galena | is present in 79 veins, or 85 per cent. |
| Sphalerite | is present in 63 veins, or 68 per cent. |
| Chalcopyrite | is present in 30 veins, or 32 per cent. |
| Arsenopyrite | is present in 27 veins, or 30 per cent. |
| Tetrahedrite | is present in 13 veins, or 14 per cent. |
| Bornite | is present in 7 veins, or 8 per cent. |
| Stibnite | is present in 3 veins, or 3 per cent. |
| Tourmaline | is present in 16 veins, or 17 per cent. |
| Rhodochrosite | is present in 16 veins, or 17 per cent. |

It is thus evident that the typical vein aggregate is pyrite, galena, and sphalerite in a gangue of quartz; that pyrite and galena exceed any other combination; and that the copper, arsenic, and antimony compounds follow considerably behind the iron, lead and zinc. The unusual gangue minerals, tourmaline and rhodochrosite, are each found in about one-sixth of the cases—rarely occurring together. Tourmaline is found in the regions where it is also abundant as a constituent of the aplite.

The occurrence of the sulphides of iron, lead, and zinc in nearly 70 per cent. of the examples used is striking. Moreover, since sphalerite, as will be seen later, is most abundant at moderate depths, it is possible that deeper development would add it to most of the additional 15 per cent. of cases in which pyrite and galena exist together. It is thus safe to say in 75 or 80 per cent. of the best-known quartz veins of aplite origin these three base metals occur in sufficiently high proportions in the solutions to be conspicuous in the resulting mineral aggregate.

The presence of copper, arsenic, and antimony is not nearly so widely evident, and in possibly half the cases these elements cannot be detected. Where they are found, they exhibit wide variation in quantity, ranging from the merest trace to a high percentage of the total vein matter. The similarity of the vein-forming solutions over wide areas is best seen in the pyrite, galena, and sphalerite; the geographic variation finds its best expression in the arsenopyrite, tetrahedrite, and stibnite. The manganese minerals and tourmaline are also geographic variables.

The general significance, therefore, of the mineral distribution may be summarized in a few sentences. The siliceous residue of the aplite stage that formed the quartz veins had essentially similar composition over wide areas, evidence of which lies in the almost universal presence of appreciable pyrite, galena, and sphalerite in the veins. Beyond this fundamental similarity, however, it varied from place to place in the amount of copper, arsenic, antimony, and manganese, these elements generally in small amounts, locally increasing sufficiently to form in their compounds a prominent proportion of the ore.

8. In the various mining districts, deep development has been prosecuted on ores of the most diverse types outlined above. Lead-rich veins, with accessory zinc; zinc-rich veins with the other base metals subordinate; even arsenic-rich veins marked by much arsenopyrite, have all been followed to considerable depths below their outcrop, and incidentally below the top of the granite. Aside from the changes in the proportion of primary sulphides, which will be discussed later, the universal trend in the composition of these veins has been an increase with depth in the proportion of quartz to ore minerals (see Fig. 19, 20, 22, 24). In mine A, for instance, the percentage of $SiO_2$ in the mill feed increased from 55 per cent. on the 400 level (about 700 ft. below the andesite) to about 75 per cent. on the 600 level. In mine B the ore shoot changes, on the 600 level from a galena sphalerite aggregate with little quartz to nearly clean quartz, and much development below has demonstrated the persistence of this siliceous phase. In mine C, 1400 ft. of development on the 1200 level, about 2000 ft. within the granite, encountered only quartz sparsely specked with pyrite, although the work is directly below a rich lead-zinc ore shoot on the upper levels. It is a striking corroboration of this tendency that the vein roots, exposed where the granite is deeply eroded, are without exception very siliceous, showing little but quartz.

9. While the total amount of sulphides in the veins thus diminishes with depth, the proportions of the different sulphide minerals themselves vary also within the vertical range of the ore shoot[33] (see Fig. 24). In certain cases the change can be followed throughout in a single mine, as in the Valley Forge, Comet, Alta, and Eva May, where exploration has traced the ore shoot through a considerable vertical range. The results may be summarized without unnecessary details of description. In each of these mines, with ores of different composition, the various minerals behave similarly. The lead content of the vein is found above, generally within 1000 ft. of the granite upper contact. Whatever zinc is present is in the main massed just below the lead zone, which places it between 1000 and 1200 ft. maximum distance from the roof. Pyrite is present throughout, but persists to greater depths than does galena or sphalerite, and is apt, below 1200 ft., to be associated with the primary copper minerals—chalcopyrite, bornite, and tetrahedrite.

[33] In this discussion of vertical zoning secondary effects are excluded. Of these, enrichment of the gold and silver, particularly the latter, is the most marked. It is probable that well-nigh all the argentite found in these veins is a secondary mineral. The greater resistance of galena to oxidation results in residual bunches in the oxidized zone, but has no influence on the occurrence of that mineral below water level. There is no evidence whatever of migration and reprecipitation of either galena, or the more soluble minerals sphalerite and chalcopyrite. Secondary enrichment is represented almost exclusively for the base metals by chalcocite, with wurtzite (of doubtful occurrence) as an occasional possibility.

The evidence obtained from neighboring veins, outcropping at different depths within the granite, corroborates the conclusions reached from these completely developed ore shoots. Consider the Eureka-Stanton group at Rimini, with outcrops ranging from 500 to 1000 ft. below the andesite capping (Fig. 13). Highest is the Eureka vein, with an ore shoot 400 ft. deep of pyrite, arsenopyrite, and galena. Similar ore in the Bunker Hill ranges between 800 and 1000 ft. within the granite. The upper workings of the Stanton, a short distance below this, show galena and sphalerite in equal amounts, with some pyrite; while in the lower tunnel little galena is to be found, with sphalerite in overwhelming excess. The zoning thus shows arsenopyrite, and galena above, and sphalerite below, all within a probable maximum distance of 1200 ft. from the batholith roof.

The numerous small mines on Big Limber Creek, near Basin, afford an equally good example of the tendency (Fig. 12). This creek heads within the andesite, and cuts down into the granite to a depth of possibly 750 ft. A depression in the old granite surface brings down the andesite again near its mouth at Basin. From north to south, therefore, the vein outcrops in this gulch progress from the andesite itself down to a maximum of 750 ft. within the granite, rising again to proximity with the contact at Basin. The northernmost vein is the Custer-Hiawatha, outcropping within the andesite. It contains tourmalinic quartz, with some arsenopyrite, and carries gold. Some galena appears on the 200 level. To the south, possibly 300 ft. below the andesite, are the Buckeye, Boston, Minneapolis, and Virginia veins. These show manganese-galena ore to a depth of about 200 ft. with sphalerite in considerable amount below this level. Still farther south, and deepest within the granite, are the Deer Lodge and Copper King veins, with chalcopyrite, bornite, and tetrahedrite. The Copper King has no galena or sphalerite but these are present in the relatively higher Deer Lodge. As the andesite cover to the south is approached, the Hope mine shows a reversion to the typical manganese-lead type of ore with much sphalerite below the 100, and some chalcopyrite on the bottom level (600).

The zoning in this district thus shows an upper belt of quartz, tourmaline, and arsenopyrite, a lead horizon between 200 and 500 ft. below the andesite, a zinc zone immediately below this, and a lean lowermost zone marked by the appearance of the copper minerals. The small vertical ranges of the different belts are probably due to the small size and slight intensity of mineralization of the veins.

Examples could be multiplied, and will appear as the different districts are described. It must suffice here to state that in all the veins known the facts corroborate the conclusions formed above: that arsenopyrite and galena are found in the higher portions of the ore shoots; that most of the zinc present occurs below the lead zone; and that this in turn

gives way in short distance to lean pyrite, accompanied by small amounts of the copper sulphides. The measure of these changes varies in different veins—most markedly between large and small veins. It can be safely said, however, that little lead is found more than 1000 ft. within the granite; that little zinc is found below 1200 ft., being mostly just below the lead zone, and that copper, while often present throughout a shoot, will in primary form predominate only in the depths.[34]

(*d*) Geographic Variation.—As they are found throughout the area of the batholith, these veins fall naturally into several groups, each group in general coincident with the extent of a so-called mining district. Between such districts veins are few and insignificant. This geographic grouping suggests a localized source for the filling of the veins of each group, a probability increased by their own similarity of composition which contrasts strongly with their general unlikeness to the mineral aggregates of other districts.

Thus the veins of the Rimini district (applying the term to the area around Red Mountain and the headwaters of Tenmile Creek) are characterized by an unusual proportion of arsenopyrite in their upper zone. This mineral, which carries some gold, has supplied the ores of numerous gold mines in the higher portions of the granite. Lower down, the veins of the region show galena and sphalerite, with comparatively little pyrite. Tourmaline is abundant in association with the earlier stages of mineralization, which include also the arsenopyrite, but is absent from some of the later lead-zinc ores.

In the Jack Mountain and Comet districts to the south, zinc is the element found in excess. Arsenopyrite is rare, and pyrite is more abundant than at Rimini. Thus, even in the uppermost shoots, such as the Alta and Gregory, much sphalerite is found with the galena, while with the scant arsenopyrite the gold value is much lessened. Below the profitable sulphides, too, the ore shoots contain much pyrite (Comet, Alta, and Eva May), in contrast to the straight quartz of the northern district.

Still farther south, in the Basin district, rhodocrosite, the manganese carbonate, is widely present, the ores otherwise resembling those of the Jack Mountain and Comet areas. This tendency is found in much greater degree in the western portions of the batholith, and in the Philipsburg, Elkhorn (on Wise River), and Butte districts the manganese minerals form a large proportion of the gangue. In the first two of these camps they are associated with the complex silver sulphides of primary source,

[34] An ore high in primary chalcopyrite will often carry enough secondary chalcocite near the surface to be mined as a copper ore, despite the presence of much galena and sphalerite below. The primary ore of such a mine is subject to the same tendencies as have been found to prevail throughout the batholith. The copper above is purely a secondary modification. The Crystal, Bertha, and Receiver mines may be listed here.

which are elsewhere rare in the vein fillings. Arsenopyrite is also abundant in these western districts. Copper, as an element in the aplite mineralization, is never abundant and is less subject to extreme fluctuation than the above metals. There is, however, a zone in the batholith, including the Wickes, Jack Mountain, Little Boulder, and possibly the Butte districts, in which chalcopyrite is found throughout the veins in somewhat greater proportion than is elsewhere usual. It is in this belt that the shallow orebodies of secondary chalcocite occur, often associated with the galena and sphalerite of the upper zones. The Crystal, Bertha, and Receiver are good examples of such seemingly inverted veins. The great primary enrichment in Butte, associated with the later quartz-porphyry, must not be confused with this type of ore deposit, which in its primary form is a lean chalcopyrite-pyrite mixture far different from the enargite, bornite, chalcocite, and covellite ores of the big copper camp.

To summarize, the geographic variation is represented by arsenopyrite and galena ores in the northern districts; by galena sphalerite, and pyrite ores further south; by manganese and silver ores along the western flank of the batholith; and by chalcopyrite-rich aggregates in a centrally located belt running from Wickes to Butte. These differences are primarily of proportion only, since in mineral composition, as we have already seen, all the veins are strikingly similar.

(*e*) Recapitulation.—In sum, therefore, the fissure veins of the aplite stage may be regarded as conforming to a widely prevailing set of conditions. They indicate, by similarity of composition, and origin closely associated with the pegmatitic after-phases of the aplite intrusions, and in occurrence, that they are so identified with the aplite as to render this source reasonable. Their position is near the outer contact of the batholith (see Fig. 22, 24), and the course and reticulated character of the fissures suggests that they represent tensional contraction cracks in the cooling mass. From these irregular openings the mineralization has diffused throughout a wider zone, and a halo of sericitic alteration marks the outer limits of the penetration. The ore minerals are in general the same in kind—galena, sphalerite, and pyrite—throughout the batholith, although in somewhat varying proportions. A vertical primary zoning of galena above, sphalerite for a short distance below, and pyrite more or less persistent throughout, is common, together with a general decrease in the proportion of sulphides with depth. The rarer minerals—arsenopyrite, chalcopyrite, rhodochrosite, tourmaline—are locally abundant, and it is in these that the geographic variation of the mineralizing solutions finds its chief expression. Arsenopyrite in the northern districts, rhodocrosite in the western, chalcopyrite in a central belt, suggest a slight regional variation in the granitic magma, of which the vein fillings are the ultimate concentrate.

### 3. *Quartz Porphyry Phase*

1. *General Description.*—The quartz-porphyry, which gives its name to this period of ore deposition, is best known in Butte,[35] although similar dikes have been found elsewhere in the batholith. As has been stated already, it shows, in the deep Butte workings, a trend toward aplitic character, together with a widening of its borders and loss of clean-cut contacts.[36] In short, it approaches, in texture and contact type, the more slowly chilled form of the true aplite intrusions. In composition the two rocks are closely allied, the quartz-porphyry being slightly more basic.

| | $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | FeO | MgO | CaO | $Na_2O$ | $K_2O$ |
|---|---|---|---|---|---|---|---|---|
| Quartz-porphyry............ | 69.9 | 15.1 | 0.4 | 0.8 | 0.6 | 1.5 | 2.7 | 6.4 |
| Aplite........ .... .. ... | 76 2 | 12 9 | 0.7 | 0.1 | 0.2 | 0.9 | 2.6 | 5.6 |

The low iron, magnesia, and lime, and the high potash, differentiate either from the normal granite or rhyolite type.

Thus the distinction, decidedly apparent on the upper levels, is primarily one of texture and acidity, both of which differences are diminishing with depth. It would seem, therefore, that the quartz-porphyry dikes were relatively basic, quickly chilled offshoots from deep-seated masses of aplitic magma which, because of size or depth, or both, were enabled to fractionalize somewhat before solidification.

In the upper levels, at least, the quartz-porphyry dikes are materially later than the aplite, as is witnessed by the cooler contact phenomena and the subsequence of the porphyry to joints common to both granite and aplite.

2. *Disseminations.*—In certain places the quartz-porphyry contains disseminated pyrite of hydrothermal origin, slightly cupriferous, and under favorable conditions of enrichment the development of secondary chalcocite is sufficient to result in a commercial ore. These deposits are in genesis strictly parallel to such better-known examples as the Ely porphyries, but in Butte are on a comparatively insignificant scale.

3. *Fissure Veins.*—The fissure-vein mineralization of the quartz-porphyry phase is, so far as known, restricted to the Butte district, where the great copper deposits are genetically referred to these intrusions. These deposits have been thoroughly described by R. H. Sales[37] and it

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[35] See Weed. *U. S. Geological Survey, Professional Paper* 74.

[36] This is strikingly shown, to name only one instance, by the Modoc dike, as now exposed from the upper levels of the Modoc mine to the lower levels of the Leonard. Above, it shows large orthoclase phenocrysts often an inch in length, at midway points it has the normal quartz-porphyry texture, with small rounded quartz-phenocrysts; below it is holocrystalline, and with difficulty distinguished from normal aplite.

[37] *Trans.* (1913), 46, 3.

is necessary only to outline here the features of significance to the general correlation of ore deposits of the batholith.[38]

1. General Description. The general situation is as follows:[39] A series of east-west fissure veins about 1000 ft. apart and 4 miles long, extends across the district, cutting alike granite, aplite, and quartz-porphyry, but cut by the Big Butte rhyolite. These veins are the earliest of the district. Cutting and displacing these are a series of northwest mineralized faults, and a later succession of northeast faults. The latter series is divided into three groups—south-dipping mineralized, north-dipping barren, and south-dipping barren respectively. The faults, particularly the northwest series, are most numerous near the eastern termination of the early east-west veins in the area now known as Anaconda Hill.

2. East-West Fissure Veins.—These strong fissures are subsequent to the aplite in age, and probably follow the quartz-porphyry also, although only their extreme eastern portions are found in contact with the latter. Their mineral filling is complex, and varies greatly along their extensive course. It can in general, however, be everywhere separated into two ages. The earlier type is essentially quartz, pyrite, and sphalerite, with small galena ore shoots near the surface, and little silver except where secondary enrichment has contributed. This stage exhibits roughly the typical primary zoning of the aplite-period fissures, galena uppermost, sphalerite below, and lean pyrite-chalcopyrite in the depths. The zones do not, however, parallel the present surface and the upper forms of mineralization occur mainly in the western and northern portions of the district with the lower pyrite zone near the surface to the south and east. This mineralization is accompanied by less rock alteration than is characteristic of the copper area.

The later vein filling occurs often as a cement for a breccia of the earlier ores, and in other instances appears to replace them. In the western part of the fissures this later mineralization is most frequently quartz, rhodocrosite, calcite, and sphalerite, with occasional high-grade silver ore shoots. In the eastern portions it consists of the copper-rich sulphides, enargite, bornite and chalcocite, with very little sphalerite. The western phase occurs with the lead-zinc ores of the upper zones of the earlier mineralization, the eastern with the quartz pyrite of the lower horizons. To the east, also, the copper mineralization is very intense,

[38] The conclusions drawn in the following description do not exactly coincide with Sales' views, especially the conception of superimposed mineral zoning of two distinct ages with the copper mineralization in the later period.

[39] See maps of vein and fault structure by R. H. Sales: Ore Deposits at Butte, Mont., *Trans.* (1913), **46**, Plates 1 and 2; also maps by W. H. Weed, in Geology and Ore Deposits of the Butte District, Montana, *U. S. Geological Survey, Professional Paper* 74 (1912).

and the early minerals are often obscure, but the undoubted priority of the great quartz-pyrite masses to the copper enrichment is witnessed both by an intervening period of brecciation and by the details of contact between the two.

There is almost no residual zinc within the "dome" of copper mineralization that thus penetrates upward into the eastern end of these fissures.[40] This fact is doubtless largely due to the low zone of early mineralization represented—a zone which was probably below the precipitating horizon of sphalerite—but on the upper levels and in outlying veins it seems likely that some sphalerite has been driven out by the copper solutions. A "halo" of zinc ore richer than the average with two generations of sphalerite frequently fringes the copper shoots in the east-west veins, and this may be partly due to such a migration and reprecipitation of the early sphalerite. This hypothesis, suggested by M. H. Gidel of Butte as long ago as 1912, seems increasingly probable with additional developments.

3. Northwest Fault Veins.—This system crosses the district from northwest to southeast, with its greatest concentration toward the east end of the earlier fissures. The faults have a nearly horizontal movement, and as suggested by F. A. Linforth, form a great shear zone, with connecting branches and intervening rhombohedral blocks. They are subsequent to the aplite and quartz-porphyry, but probably prior to the Big Butte rhyolite.

Their mineralization is most intense at the southerly end of the system, where the copper "dome" in the east-west veins is found, and this may be reasonably regarded as an extension of the doming phenomena into the later fractures.[41] Most significant is the comparative absence of the massive quartz-pyrite type of ore from these faults, an absence that must be regarded as corroborating the evidence already given of its priority to the copper mineralization. In respect to the copper minerals themselves, however, there is little distinction between the fault veins and the earlier fissures. Possibly less enargite, more bornite, and more chalcocite characterize the typical aggregate in the later ores.[42] Sphalerite is found in the fault veins, in zones peripheral to the copper, and probably represents largely the original zinc content of these solutions, carried to remoter

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[40] Ray claims to have found some residual zinc among the copper minerals, but such occurrences, while frequent on the border of the copper zone, must be very rare within it.

[41] In Fig. 25 the range lines of the ore shoots in the Northwest Vein Series are arranged to show the "doming" in the copper mineralization of these veins. The veins on the edges are veins near the borders of the copper area of mineralization.

[42] See Sales, *Trans.* (1913), 46, 3, for detailed discussion of these variations in the copper solutions.

and cooler regions before precipitation.[43] It is certainly a later generation than the residual sphalerite in the older ores, although Gidel's migration hypothesis might explain even this without necessitating the presence of zinc in the copper mineralization. Like the later ores of the western area, this zinc is accompanied by manganese, and on the whole we prefer to attribute both occurrences to peripheral phenomena of the copper "dome."

4. Northeast Fault Veins.—The mineralized fault veins of northeast strike are so intimately associated with the northwest system as to strongly suggest mechanical resultants of that movement. Their age, also, must be nearly the same, for while normally they cut and displace the northwest faults it is not infrequent for renewed movement on the latter to reverse this relation. The mineralization is nearly identical, but is less abundant in the later fractures, which mark its waning and final cessation. All the minerals of the earliest copper veins are repeated in these, the latest and last, but chalcocite is increasingly predominant and enargite insignificant.

5. *Summary.*—This is the briefest possible recapitulation of the bare facts. Interpretations of their genetic significance must and do vary, and we can here only suggest one that seems to fit the internal evidence of the district as well as the general tendencies of the batholith.

In the first place, the early mineralization of the east-west veins, which is known to slightly antecede the copper period,[44] may safely be correlated with the aplite type so widely distributed throughout the granite mass. In mineral composition, in reference to the granite surface[45] and in primary zoning, it possesses the requisite characters. It seems clear that at this stage no unusual sources of mineralization had yet been tapped, although quite possibly the quartz-porphyry dikes had already penetrated upward to their present position.

Structural evidence within the veins seems, in the second place, to postulate a slight pause after the conclusion of this period—possibly representing the delay while new disturbances of the quartz-porphyry reservoirs released the great copper-bearing solutions that now began to migrate into the east end of the fissure system. At their earliest stage these solutions seem to have carried all the elements that enter into the

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[43] Sphalerite has been shown in the laboratory to precipitate at lower temperature than the copper sulphides.

[44] This idea is also held by W. H. Weed, *Trans.* (1903), **33**, 747.

"The older primary quartz-pyrite veins were reopened by later movements, correlated with a period of volcanic activity; and were penetrated by hot alkaline waters carrying copper and arsenic in solution."

[45] The projection of the granite cover over the Butte district involves more than the usual difficulties, but it can nevertheless be approximated as the flat upper contact of the granite is known to the southeast, south, west, and northwest, and roof pendants or large inclusions of andesite are found within the productive area itself. Such foreign rocks are never found elsewhere very far from the granite contact.

composition of the many copper-rich sulphides that characterize the Butte district, so that from the first, enargite, tennantite, tetrahedrite, bornite, covellite, and chalcocite were formed in the veins. Intense rock alteration, marked by sericitization, pyritization, and silicification, also marked the copper period from the beginning. Throughout the formation of the northwest and northeast faults, this mineralization continued, undergoing the gradual transitions already outlined, but without any real cessation until its final stop, probably just before the rhyolite eruptions to the west.

This long-extended period of ore deposition we attribute to the contribution of deep-seated reservoirs, probably fractional residues of the quartz-porphyry period—the quartz-porphyry itself, it will be remembered, being merely a partially differentiated, slow-cooling aplite. Such deep sources had the advantage of further fractionalization than the more superficial residues of the aplite stage, and they contained a higher concentration of metallic contents. It is doubtful, however, whether this alone is adequate to explain the great richness and amount of the copper deposits, and it seems that possibly an originally high copper content of the local portion of the granite magma is required in addition. The copper-rich belt that has been mentioned as crossing the batholith from Wickes to Butte may be a significant guide along this line of speculation.

As might be expected with the above interpretation of the quartz-porphyry and the mineralization, the ores of Butte are found deeper within the batholith than in the case of the true aplite-derived veins. These, as we have seen, carry their lead and zinc not lower than 1000 or 1200 ft. within the granite. These depths will also include the upper zones of the early Butte mineralization. The shallow mines of the western and northern sections of the camp can hardly have penetrated that far into the batholith, since its cover is now found to the north, west, and south at elevations lower than many of the shaft-collars, and the contact is everywhere nearly flat.

The copper ores, however—a primary enrichment by solutions from a deeper reservoir at a later period—found the precipitating conditions at lower points, so that they range from 1000 to 1200 ft. within the granite at their outcrop to 4000 ft. as now developed, and show no signs of an approaching bottom. A suggestion that this great vertical range is due to the long continuance of the mineralization, and the downward migration of the precipitating point, is given by the tendency of the later ore shoots, as in the northwest faults, to begin at lower elevations than is the case in the east-west veins[46] (see Fig. 25).

[46] This tendency is also graphically shown by contrast of the longitudinal sections shown in Plate 5 and Fig. 6 with Fig. 6*a* in Ore Deposits at Butte, Mont., by R. H. Sales, *Trans.* (1913), 46, 36, 74.

The only other deep ore deposit of the batholith, the Granite-Bimetallic at Philipsburg, also shows evidences of later primary reënrichment. (See Fig. 11, 20, 24.) Of this ore W. H. Emmons writes as follows:[47]

"The paragenesis indicated above suggests that the ore first deposited consisted mainly of quartz, rhodochrosite, calcite, pyrite, arsenopyrite, stibnite, tetrahedrite, tennantite, galena, and zinc blende. Pyrargyrite, proustite, realgar, and orpiment are present in this ore in smaller amount, and it is possible that they have formed at the same time. After the first deposition of the ore the veins at some places were fractured and reopened. Some of these fractures were parallel to the walls and others cut across the veins and produced slight faults in the bands of vein quartz.

"Solutions more highly carbonated, presumably introduced from below, deposited much rhodochrosite, some calcite, and quartz, with a minor quantity of the sulphides, . . . . . ."

In the above statement, the earlier ore is seen to be, aside from the listing of the rare minerals, essentially similar to the early Butte ores, and to that of the aplite veins in general. The later solutions are entirely different, and it is probable that wherever found these, the products of an extra fractionalization, carry to an extreme point the original peculiarities of the magma in that region.

## C. Rhyolite Period

### 1. *Early Rhyolite Phase*

1. *General Description.*—The early rhyolite is a rock widely prevalent over the northern end of the batholith. It is considerably later in age than the granite, having flowed out on an eroded surface of that rock. It is also subsequent to the Eocene peneplain, with which its lower contact often coincides, but is prior to the inner valleys and the later Tertiary gravels. The north-south normal faults occur in the rhyolite as well as the earlier formations. Probably Oligocene age most nearly fits these conditions.

The magmatic waters of these rhyolites contain typically only silica and iron; and quartz and pyrite, either in disseminations or veinlets, form the bulk of their mineral deposits. Accessory gold furnishes the commercial element, and mechanical enrichment by descending waters is generally required to raise this to a profitable grade.

2. *Disseminations.*—The best examples of disseminated ores are the mines strung along the Tenmile-Basin Creek Divide south of Helena (see Fig. 13). They exploit low-grade gold ore in the early rhyolite dikes and flows that cap this ridge. The Porphyry Dike may be taken as a type.

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[47] Emmons and Calkins: *Op. cit.*, 177.

The rhyolite here consists of a succession of alternating breccia, ash beds, and porphyritic flows with an intrusive dike along the western edge of the series. The effusive rocks lie upon a somewhat irregular granite floor, which must, by the basic character of that rock, be nearly identical with its former contact surface. Faults of the north-south Continental type cut and displace all the rock series.

The gold is primarily associated with finely disseminated pyrite in the fine-grained intrusive rhyolite and has been enriched along the vertical joints of this dike. The breccia and ash beds are practically barren, but the flows carry some gold. Samples along the faults are very erratic, suggesting mechanical transportation of the precious metal, with impoverishment in some places and enrichment in others. Practically the entire gold content can be recovered by amalgamation, which suggests that the native metal itself must be a primary mineral of the rock. This rhyolite gold is low-grade, running from $7 to $11 per ounce, and can, even in the placers, be readily distinguished from the $16 or $18 gold of the granite period veins.

3. *Impregnations.*—This term covers the instances where the ore deposit has been formed by the penetration and diffusion of magmatic waters into and through portions of the rhyolite. The Woodrow Wilson, south of Rimini, is a good example. The ore here is a rhyolite of the early period intensely silicified and pyritized, and traversed by numerous quartz-pyrite (largely oxidized) stringers. A certain amount of gold accompanies this alteration and mineralization. The occurrence differs from the Porphyry Dike in the evidence of hydrothermal action, and the contribution of silica and iron to the rock. The source of these additions is probably the deeper portion of the same dike.

4. *Contact Deposits.*—The only occurrence of contact mineralization at the edge of the early rhyolite is the Monte Cristo mine in the Rimini district. Here an orebody has formed on the wall of a rhyolite mass intrusive into the granite. The ore consists of specular hematite and cuprite, with some very rich silver. Development has been shallow, so that no evidence of the character or persistence of the deep-seated ore is available.

### 2. *Dacite (or Later Rhyolite) Phase*

1. *General Description.*—These rocks are readily placed as to age, since their ash and tuff phases are intercalated with the Pliocene gravels of the Tertiary valleys. They are thus long subsequent to the cooling of the granite, and were extruded upon a surface essentially in its present form. In some localities, notably north of Butte, these dacites have piled up to a thickness of over 1000 ft., with individual flows ranging from basic andesitic types to quartz-porphyries and true rhyolites. Some quartz is almost always present, however, and biotite is very char-

acteristic of all the phases. In a general way the upper flows are more acidic than the lower ones, and this tendency is seen also in the intrusive areas, where more acid types penetrate earlier more basic masses.

2. *Fissure Veins.*—This dacite phase lacks, in its ore deposits, the disseminations, segregations, and contact phenomena of the more energetic igneous periods, and its production is confined largely to fissure veins, filled by magmatic solutions. The mineral filling is almost always limited to quartz and pyrite, lacking the sulphides of the rarer metals. Much of the silica is amorphous and chalcedonic, and the frequent calcite increases the impression of cool and shallow conditions of deposition.

The best examples of this type are the mines of Lowland Creek, the Ruby, Kit Carson, Memphis, and Gopher. The first-named has been well described by Knopf.[48]

"The ore-bearing zone extends from the main shaft to the Columbia claim, a distance of several thousand feet, the general trend being S 20° E. Within this zone the ore occurs in shoots or more properly in parallel veins . . . disposed roughly in echelon fashion. . . . The ore consists of angular dacite fragments cemented by quartz, calcite, and minor adularia. The adularia, where embedded in quartz and calcite is not easily distinguishable. . . . The quartz is commonly clear, glassy, and drusy, but where it is solid it is of compact saccharoidal texture. Some of the siliceous veinlets show a porcelain-like texture, but such cryptocrystalline quartz is far less common at the Ruby mine than at the surrounding properties. The sulphides, which are confined to the gangue material that cements the dacite fragments, comprise pyrite, argentite, and possibly others. . . ."

This mineralization is typical of many such fissures and shear zones in the later rhyolite, wherever they are found.[49]

### 3. *Summary*

Thus in both rhyolite phases the normal mineralization is generally restricted to the commoner elements, silica, lime and iron, contributed to the cooled intrusions from deeper-seated portions of the same material. As might be expected from the slight erosion that has taken place since their occurrence, the rhyolitic deposits have the characters suggesting formation at shallow depths. Knopf summarizes their character as follows:[50]

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[48] Knopf: *Op. cit.*, 126.

[49] Knopf places many of the orebodies of the Clancy district in this dacite period. Among these are the King Solomon, Free, Coinage, Fleming, etc. From our own investigations we doubt whether, in some of these cases, the dacite has done more than enrich preëxisting shoots, so that in want of conclusive evidence we will leave these occurrences out of the discussions.

[50] Knopf: *Op. cit.*, 55.

"The orebodies are mainly fissure veins of branching and irregular character. . . . As a rule, the orebodies carry subordinate quantities of sulphides, and are worked for their content of precious metal alone. A prominent feature of the deposits is the prevalence of crypto-crystalline quartz in the gangue material. The crypto-crystalline quartz appears in three varieties: (1) A dark gray blue flinty variety; (2) a dense-grained white variety resembling porcelain; and (3) chalcedony. These three modifications, however, are likely to be present together in the same deposit, confusedly intergrown and each variety grading into the other. . . . .

"The metasomatic alteration of the wall rocks of the late Tertiary veins is principally in the nature of a thorough sericitization, accompanied by the introduction of carbonates and locally by the development of chlorite."

These statements of Knopf are in essential accord with our own observations of ore deposits of the two rhyolite periods, although more applicable to the later dacite stage.

The characters found are those approached by the weak dying phases of other periods of mineralization, and these deposits are unique in the absence of the stronger hot temperature features rather than in the possession of the chalcedonic quartz and calcite.

## PART 3.—CONCLUSIONS

### A. Association of Ores and Igneous Rocks

#### 1. *Magmatic Source of Vein Filling*

The ores are associated with intrusive rather than extrusive forms of igneous rocks. Exceptions are Baggs Creek, Porphyry Dike (partly), Emery (partly), Evening Star, and Elkhorn iron mines, none of which are very important. In the cases of Emery and Evening Star their position is due to the favorable cavities in flows, with a probable source below. Intrusive phases have produced at least 99 per cent. of the mineral wealth in the area described. Therefore, mineralization depends upon properties held exclusively by intrusives, which include: (*a*) the chance for differentiation and fractional crystallization, with the accumulation of magmatic residues in deep-seated reservoirs; (*b*) the development of fissures, by shrinkage, which tap such reservoirs. Hence we conclude that magmatic waters have been the predominant source of ore deposition in every igneous period. We see no reason why meteoric waters should thus favor dense crystalline intrusives over bedded breccias and tuffs and porous flows. Against lateral secretion is the undoubted fact that the original disseminated deposits, such as Red Rock, Heddleston, and Porphyry Dike, have remained disseminated, with no primary

concentration into veins. Acceptance of a plutonic magmatic source for the ore deposits seems quite unescapable.

### 2. *Intrusive Wall Rocks*

It follows that the igneous periods which are most largely of intrusive type have the most mineralization. Hence the granite period leads the others while the andesite comes ahead of the rhyolite.

### 3. *Ores and Successive Igneous Phases.*

Within an igneous period, the successive phases have increasing proportions of vein formation, which suggests the dependence of this upon magmatic differentiation. Thus, in the granite period the ores of the granite phase are neither proportionately as numerous nor as important as those of the aplite phase, which has only 10 per cent. or less of its extent, while the very small quartz-porphyry phase carries a relatively large and rich mineralization.

## B. Progression of Mineralization

### 1. *Common Order*

In each phase the mineralization follows a common order: first, the products derived from the outer portion of the intrusive, found as border disseminations and differentiations and as contact deposits; second, primary segregations within the mass of the rock; and third, contribution from deep-seated reservoirs to fissures and faults.

These have connecting links. The border differentiations are of the basic and quickly crystallizing elements, while the internal segregations are a siliceous and metal-bearing residue from slower cooling. These latter in turn are analogous to the sources of the fissure mineralization, and represent probably the smaller reservoirs that were not tapped by any channels of circulation. Thus the whole succession follows the accepted laws of the crystallization of igneous magmas.

### 2. *Common Variation in Character*

The character of vein filling also shows a progressive change within the limits of an igneous period. The earliest ores, border segregations and contact deposits, are the highest in iron; and magnetite, hematite, pyrrhotite, pyrite, and chalcopyrite are found with little accompanying quartz. The first fissure veins most nearly approach the border types of mineralization. Thus, in the early granite fissures, quartz and pyrite

with accessory gold are the predominant minerals, the pyrite often in excess. In the later stages, as at Scratch Gravel and Marysville, lead, with its accompanying silver, becomes more important, while the iron content has decreased appreciably.[51] Similarly, in the aplite phase the early ores are tourmaline, quartz, and pyrite with gold, following the type of the pegmatite segregations. The base metal sulphides, with accompanying silver, appear in the later vein filling. In the quartz-porphyry phase the long-continued mineralization shows progressively decreasing iron and increasing copper in the solutions while gold is present only in the most minute quantity. All the phases in their dying stages show a highly siliceous type with little sulphide content, and with chalcedony and calcite in some amount.[52]

## C. Relation of Ore Deposits to Forms of Intrusion

### 1. *Contact Deposits*

The border deposits, included in the first of the above divisions, can generally be referred to certain forms of the intrusive rock. They particularly favor protruding cupolas or peninsulas with their facilities for rapid cooling. Such masses both contain the great proportion of disseminations and segregations and have most often contributed the solutions to outlying contact deposits. Small dikes, on the other hand, are of little value in this respect, possibly because too quick chilling has prevented the segregation or escape of the magmatic waters. Where the intrusive mass is of laccolithic form, as in the Golden Sunlight mine, the upper contact is the locus of mineralization, while the lower one is comparatively barren. Thus gravity appears to occupy a subordinate place in fixing the position of such ore deposits except where, as in basic flows, the elements of pneumatolysis and magmatic water pressures are negligible.

### 2. *Segregations*

As has been stated above, the deep-seated segregations are acid, and are apt to contain such high-temperature minerals as tourmaline, hübner-

[51] In fault veins later than the fissure veins of the Empire mine, Marysville district, silver is the important ore mineral and gold has ceased to be an appreciable constituent of the mineralizing solutions.

[52] This progression in type suggests rather wide possibilities of correlation with the parent magma. The early ore deposits are derived immediately from the borders of the intrusive rock, which is in the great majority of instances more basic and higher in iron than the deeper-seated portions. Hence both segregations and mineral residues will reflect in their composition this feature of their source. The later deposits, such as fissures penetrating through the chilled crust and tapping deeper portions of the magma, will by their decreased iron indicate the relatively greater acidity of these regions, not affected by the basic contact differentiation. Thus the orebodies seem to reflect the variation in the igneous mass, slowly cooling and crystallizing from its surface toward its interior.

ite, molybdenite, and cassiterite. They are most frequent in regions where fissures are rare, and are generally of such small size as to have been readily missed by even abundant fissuring.

### 3. *Fissure Veins*

The fissure ore deposits are generally situated above the high points of their accompanying igneous intrusions (see Fig. 23). Thus the granite fissures top the outlying cupolas of the batholith, and are absent from its central portion, where erosion has obliterated these protruding masses. The aplite and quartz-porphyry fissures, likewise, are above the main bulk of these rocks, accompanying dikes which have probably pushed upward from larger masses in depth. This is particularly evident in the case of the Butte quartz-porphyry. The undoubted fact may be due to several interacting causes. (*a*) If preëxisting fissures were available, the igneous rock would push ahead along these lines of weakness. (*b*) Such local projections would cool quickly and form internal fissures. (*c*) These, and the preëxisting fractures, would become convenient outlets for the metalliferous and siliceous residues of the slower cooling magma below. In any case, strong upward pressure for the solution is necessitated.

## D. Geographic Variation

In the case of the aplite fissures the geographic variation of the vein filling has been described. Some such differences are also apparent with the ore deposits of other phases. Thus the contact ores of the granite phase are predominantly of the iron-gold type to the west but are relatively high in copper to the south. The granite fissures show little variation, although somewhat more apt to contain lead and zinc along the northern and eastern borders of the batholith than elsewhere. With the fissures of the aplite stage, provinces of zinc-rich, lead-rich, arsenic-rich ores can be readily distinguished. Still greater variability is suggested by later mineralizations which in Butte bring in the unusual copper-rich sulphides, enargite, bornite, chalcocite, etc., while in Philipsburg complex silver minerals were introduced.

Since all these ore deposits are of undoubted primary origin with sources within the igneous bodies in which they occur, such variations can be referred only to differences between the contents of the reservoirs. As these are the end products of a series of fractional crystallizations, the varying proportions of the different elements are in turn carried back to an original irregularity in their distribution in the parent magma. The facts are in further accord with this conception. The earlier veins, products of the first crystallization, are far less diversified than those formed after the later stages, and in the latest ores of all, as we have

already seen, the variation is most pronounced. If the metallic contents of the original magma were left behind and concentrated during a succession of partial crystallizations, the geographic variation would be accentuated in the later phases of vein formation. This is, in fact, the case.

## E. Factors Governing Vertical Distribution of Ore Shoots

### 1. *Mineral Ranges*

Within the veins themselves certain tendencies are widespread. Most noticeable is the decrease, with depth, of the proportion of sulphides to quartz (see Fig. 17, 18, 19, 20, 21, 22, 24, 25). This is true of all the veins of all periods.[53] In mineralization of long duration, as in Butte, there is greater range of sulphide precipitation than in any of the more restricted phases (Fig. 25). Of the sulphides, pyrite exhibits the greatest range, and is generally found from top to bottom of the veins. Galena, also widely distributed, seldom occurs in the lower portions, but travels far in the cooler solutions of the upper zones before precipitating. Its range is essentially that of arsenopyrite also. Sphalerite is a close associate of galena, but its greater portion comes down at points below the horizon of greatest galena precipitation to which it forms a sub-zone of comparatively shallow depth. The copper minerals are generally associated with the deeper horizons, and appear in greatest amount below the galena and sphalerite zones.

The manganese minerals, rhodocrosite and rhodonite, are oftenest found well toward the top of the mineralization. Some elements are found in different forms at different horizons: Scheelite is associated with the upper gold-pyrite ores of the granite fissures while hübnerite occurs with galena and sphalerite in the central zones of the aplite veins; stibnite and arsenopyrite are found at or above the lead-zinc horizon, while tetrahedrite and tennantite accompany the deeper copper ores.

Too definite boundaries cannot be assigned to these horizons, and since dying phases of mineralization, or later renewals, may encounter precipitating conditions at lower levels than the first energetic outbursts,

[53] In the longitudinal sections (Fig. 18, 19, 20) the largest orebodies of each stage of mineralization, in which the bottoms of the richer ore shoots are approximately known, were selected. The sections show the decrease in the grade of the ore by the smaller sizes of the stopes at each successive level in depth, and in the case of the Drumlummon mine, an absolute absence of minable ore is shown on the lowest level. The larger the shoot of ore is laterally, the greater its vertical dimension usually is, so that the conclusions so forcibly shown by the longitudinal sections of the largest mines of each type may be applied without reservation to the smaller ore shoots of the same types. The ore shoots of the Hecla mines and the Elkhorn mine are also smaller and leaner on the lower levels. In Butte this tendency has not yet become manifest in the primary ore shoots, either of the zinc or copper type.

there will necessarily be much overlapping and migrating downward of zones. For a given stage of mineralization, however, the succession outlined above and recapitulated below seems to be universal and to rely on fundamental causes.[54]

| | Top Much Sulphide Bottom Little Sulphide |
|---|---|
| Stibnite | ......... ...... |
| Rhodocrosite | .................. |
| Arsenopyrite* | ................... |
| Galena | ................... . |
| Sphalerite | ...................... |
| Chalcopyrite | ...................... |
| Tetrahedrite | ...................... |
| Bornite | ...................... |
| Pyrite | .................................. |
| Quartz | ..................................... |

* See concluding discussion, p. 367.

[54] The idea of a primary zonal arrangement in the metallic sulphides deposited from hot ascending solutions is not a new one. Several writers on mining geology have mentioned the chemical probability, and many instances are cited in the literature on ore deposits. A series of relative solubilities for metallic carbonates has been established by R. C. Wells and others, and a series for the orderly precipitation of the sulphide minerals from cold acid solutions has been determined by E. Schuerman. With the exception, however, of a statement by F. W. Clarke that sulphides of tin, arsenic, and antimony are more soluble than other sulphide ores in alkaline sulphide solutions, we have found no reference to this group, most important in the consideration of primary zoning of ores.

(*A*) W. H. Weed: Ore Deposition and Vein Enrichment by Ascending Hot Waters, *Trans.* (1903), **33**, 751.

"It is well known that the metallic sulphides are soluble in alkaline solutions under heat and pressure. . . The more soluble substances will be carried further upward before precipitation, and one might even suppose, if the solubilities of the substances were sufficiently unlike, that zones would be found each one of which consisted mainly of the particular substance thrown out by the change of pressure. This would produce an orderly distribution of the ores in a vertical direction. . . In the writers' own experience (Geology of the Castle Mountain Mining Districts, Montana, *U. S. Geological Survey, Bulletin* 139 (1896). Geology of the Little Belt Mountains, *U. S. Geological Survey* (1900), *20th Annual Report,* Part III, 271–461) the order appears to be galena on top, passing into highly zinciferous ores below, and this into low-grade pyrite."

(*B*) *Types of Ore Deposits,* edited by H. F. Bain, 347. San Francisco, *Mining and Scientific Press,* pub., 1911.

"An apparent, but deceptive, relation of the nature of the ore to country rock may sometimes be due to the fact that, in solutions rising in a fissure, as the pressure and heat diminish, certain ores may be deposited at a certain depth and others at another depth, according to the conditions of pressure and temperature that permit the precipitation of each."

(*C*) Beyschlag, Vogt, and Krusch: *Ore Deposits,* **1**, 211.

"Since pressure and temperature, which are regarded as causing these variations in primary deposition, generally promote the solubility of substances, those then which are the most easily dissolved would separate the less readily from solution and would consequently be found in the neighborhood of the surface. This is probably

The evidences of this, for a particular phase in particular districts, are seen on the accompanying zoning diagrams (Figs. 21, 22, 24).

Wherever the tops of ore zones are visible in several veins in a district, it is apparent that all the horizons reach higher levels in large than in small veins. Each zone is also apt to cover a greater vertical range in the large fissures.

---

the reason why minerals which form many easily soluble compounds are found in the upper levels. Again, differences in the filling of deposits, such as may be referred to change of temperature and pressure, may make themselves evident not only in the particular ores deposited, but also in the internal structure of the deposit."

(*D*) Such vertical arrangements of minerals due to primary mineralization have been noted by others in widely separated districts. A brief list of some of the better known occurrences follows, the order of naming the predominant minerals being from the top downward.

1. Change in character of mineralization:

(*a*) Castle Mountains, Montana, W. H. Weed: galena; sphalerite; low-grade pyrite.

(*b*) Little Belt Mountains, Montana, W. H. Weed: galena; sphalerite; low-grade pyrite.

(*c*) Neihart, Montana, R. H. Sales: galena; sphalerite and pyrite; quartz and carbonates of iron, calcium, and manganese.

(*d*) Dolcoath mine, Cornwall, C. Le N. Foster: copper to 1,000 ft., predominately in slates; tin below, chiefly in granite.

(*e*) Clausthal district, Germany, Vogt: galena; sphalerite.

(*f*) Berg district, Germany, Vogt: galena; sphalerite; siderite.

(*g*) Pyrenees Mountains near Angeléze Gazost, Vogt: galena; sphalerite.

(*h*) Wood River district, Idaho, Lindgren: silver; galena; sphalerite; pyrite and quartz.

(*i*) Slocan district, B. C., Lindgren: rich silver, galena, sphalerite; poorer ores with more siderite, pyrite, and quartz.

(*j*) Coeur D'Alene district, Idaho, Ransome: rich silver; galena; poorer ores with more pyrrhotite, pyrite, and sphalerite.

(*k*) Przibram district, Bohemia, J. Schmid: galena, sphalerite, chalcopyrite, pyrite, and arsenopyrite, with a gangue of calcite, siderite, and quartz; quartz and sphalerite increase proportionately.

(*l*) Hualpai district, Arizona, F. C. Schrader: rich silver, galena, sphalerite, chalcopyrite, and pyrite; galena decreases and chalcopyrite increases to 700 ft. depth.

(*m*) Freiberg, Saxony, Alte Hoffnung Gottes Mine, R. Beck: galena decreases until at 1,650 ft. it is almost entirely replaced by sphalerite and pyrite. The silver content of the galena has also decreased.

(*n*) Bingham Canyon, Utah, Perry, Locke, and Bateman: relative to igneous rock, outer zone has predominant galena; median zone sphalerite; and inner zone pyrite and chalcopyrite.

(*o*) Bersbo mine, Sweden, Sjoengren: copper decreases and zinc increases in depth. (This is a contact deposit, and the inversion of the usual relation may be due to the position of the ore shoot relative to the igneous contact.)

(*p*) Sulphur Bank, California, W. H. Weed: sulphur is found in the upper 200 ft., giving place to quicksilver ores below that level.

(*q*) Boulder County, Colorado, Lindgren: association of scheelite and ferberite with gold-telluride ores noted at shallow end of zone, giving place to the sphalerite, galena, chalcopyrite, pyrite ores of Leadville at the deeper end.

### 2. *Factor of Progressive Cooling*

A comparison of the vertical position of ore shoots in the different phases of mineralization included in the granite period shows downward migration, with the lapse of time, of the critical conditions of precipitation. This is well seen in Fig. 25. Thus it is evident that while solubility, pressure, and chemical reactions must have borne important parts in the deposition of ore minerals, the factor that determined the general position of the shoots was the critical temperature of precipitation, modified, however, by the other conditions. In addition to lowering the horizon of precipitation to points further within the igneous rock, the progressive cooling and crystallization of the batholith necessitated the derivation of the vein-forming solutions from greater and greater depths. Thus we have both elements essential for the repeated phases of mineralization as they are now found—sources progressively lower in the batholith, and a horizon of precipitation at lower levels with each stage.

## F. Summary

Such are the conclusions that have resulted from the correlation and genetic classification of the ore deposits of the Boulder batholith. Over 500 mines, including most of the important producers of the region, have been personally examined and the entire area of the batholith has been geologically mapped. The ideas of ore deposition outlined above have evolved gradually during the accumulation of data, and have finally thus crystallized. Information has subsequently been obtained of new districts and mines, and this has without exception confirmed the conclusions already arrived at.

It has so happened, also, that both authors have recently seen much of the great Idaho batholith, and many of the theories evolved in Montana have proven applicable to this neighboring granite area, especially as regards the gold-pyrite veins.

A brief recapitulation of old conclusions confirmed and new ones reached may fitly close this discussion.

---

2. Decrease in metallic content in depth without change of type of mineralization:

(*a*) Rio Tinto mine, Spain, Vogt and Beck: decrease of chalcopyrite in depth.

(*b*) Tin mines of Bolivia, Vogt: stanniferous pyrite becomes poorer in depth.

(*c*) Falu mine, Sweden, Sjoengren: decrease in copper and silver in depth.

(*d*) Sala mine, Sweden, Sjoengren: galena decreases in depth to 100-ft. level; silver decreases more rapidly than galena.

(*e*) Empire district, Colorado, Spurr and Garrey: chalcopyrite decreases in depth more rapidly than pyrite.

(*f*) Freeland group, Empire district, Colorado, Spurr: galena, siderite, and rhodocrosite decrease on lower levels.

(*g*) Siliceous lead ores in limestone, Weed: decrease in galena and silver in depth.

1. The ores are associated with intrusive rather than extrusive forms of igneous rocks.

2. The most varied types and largest number of ore deposits are associated with the periods of greatest intrusive action.

3. Within an igneous period, the successive magmatic differentiates furnish increasing proportions of vein-forming solutions.

4. The time sequence of ore deposition in any igneous phase follows a definite order: first, contact and border; second, internal segregations; and third, fissure and fault veins filled from deep-seated sources.

5. The character of vein filling shows progressive change within the limits of an igneous period, the main feature being a decrease in the proportion of iron.

6. Projecting and quickly cooled masses of the igneous rocks, which favor moderately rapid crystallization, and ejection of mineralizers, are the general loci of border types of ore deposits.

7. Deep-seated segregations are more acid than peripheral segregations.

8. The fissure ore deposits are situated in and above the high points of their accompanying igneous intrusions.

9. There is a geographic variation in the composition of vein filling, which increases with successive phases of a period of mineralization.

10. A definite vertical primary zoning exists in the fissure veins.

11. The horizon of ore precipitation migrates downward through successive phases of mineralization.

All in all, it is evident that the ore deposits are essentially a superficial phenomenon, connected fundamentally with crustal conditions in the batholith, and only found in the depths where these conditions are accidentally repeated.

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## APPENDIX B

*A. Andesite Period.*

All of the types of ore deposits described in this paper are found in the following districts:

1. Emery.
2. Elliston.
3. S. Baldy.*
4. Elkhorn.
5. Radersburg.*

*B. Granite Period.*

(*a*) Segregations and disseminations are found in the following mining districts:

1. Helena.
2. Sheridan.
3. Elkhorn.
4. Whitehall.
5. Heddleston.
6. Red Rock Creek.

(*b*) Contact replacements are found in the districts listed below:

1. French Gulch.
2. Highland.

* See footnote on page 337.

3. Argenta.
4. Bannack.
5. Blue Wing.
6. Philipsburg.
7. Blue Eyed Nellie
8. Vipond.
9. Red Lion.
10. Georgetown.
11. Utopia.
12. Polaris.
13. Whitehall.
14. Moose Creek.
15. Melrose.

(c) The following are the districts characterized by fissure veins of the Granite Phase:

1. Pony.
2. Garnet.
3. Scratch Gravel.
4. Marysville.
5. Gould.
6. Heddleston.
7. Racetrack.
8. Pikes Peak.
9. Georgetown.
10. Norris.
11. Mammoth.
12. Unionville.
13. Argenta.
14. Elkhorn (S).*
15. Polaris.
16. Melrose.
17. French Gulch.
18. Confederate Gulch.
19. Colima.
20. Top O'Deep.
21. Clinton.

Probably also the following districts belong to this group.

22. Winston-McClellan Creek.
23. Beaver Creek.
24. Indian Creek.
25. Radersburg.*
26. Bannack.
27. Tidal Wave.

And scattered individual deposits of other districts.

2. *Aplite Phase.*

(a) Segregations and Disseminations.

1. Red Lion District.
2. Basin District.
3. Rimini.

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* See footnote on page 337.

(b) Contact Replacements.*

1. Bryant.*
2. Elkhorn.*
3. Philipsburg.*

(c) Fissures.

1. Helena.
2. Rimini.
3. Elliston.
4. Jack Mt.
5. Basin.
6. Comet.
7. Wickes.
8. Prickly Pear.
9. Little Boulder.
10. Warm Springs Creek.*
11. Philipsburg.*
12. Elkhorn (S).
13. Oro Fino.
14. Butte.*
15. French Gulch.
16. Clancy.
17. Peterson Creek.

3. *Quartz-Porphyry Phase.*

(a and b) No segregations, disseminations, or contact replacements of importance.

(c) Fissures. Several ages of fissuring connected with the mineralization of this phase at Butte. Possibly also represented in the later reopening of the Granite Bi-Metallic Vein at Philipsburg. No other indications of this type of mineralization yet discovered.

C. *Rhyolite Period.*

1. *Early Rhyolite Phase of Mineralization.*

(a) Disseminations, contact, and fissure impregnation found only near Rimini.

2. *Dacite Phase.*

(a) Contacts and Fissures.

1. Clancy.
2. Warm Springs Creek.*
3. Lowland Creek.

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* In the above lists the asterisks mark doubtful cases, the S. Baldy district being marked because the age of this andesite is not known; the Radersburg district because while the veins are entirely within andesite the solutions may have had their source in the granite; the aplite contact replacements at Elkhorn, Hecla, and Philipsburg because the only criteria for their classification is the character of their vein fillings; the Butte district under aplite fissures because these fissures may be of the quartz-porphyry phase of mineralization entirely; and the Warm Springs Creek district under rhyolite fissures because of some uncertainty as to the age of the veins.

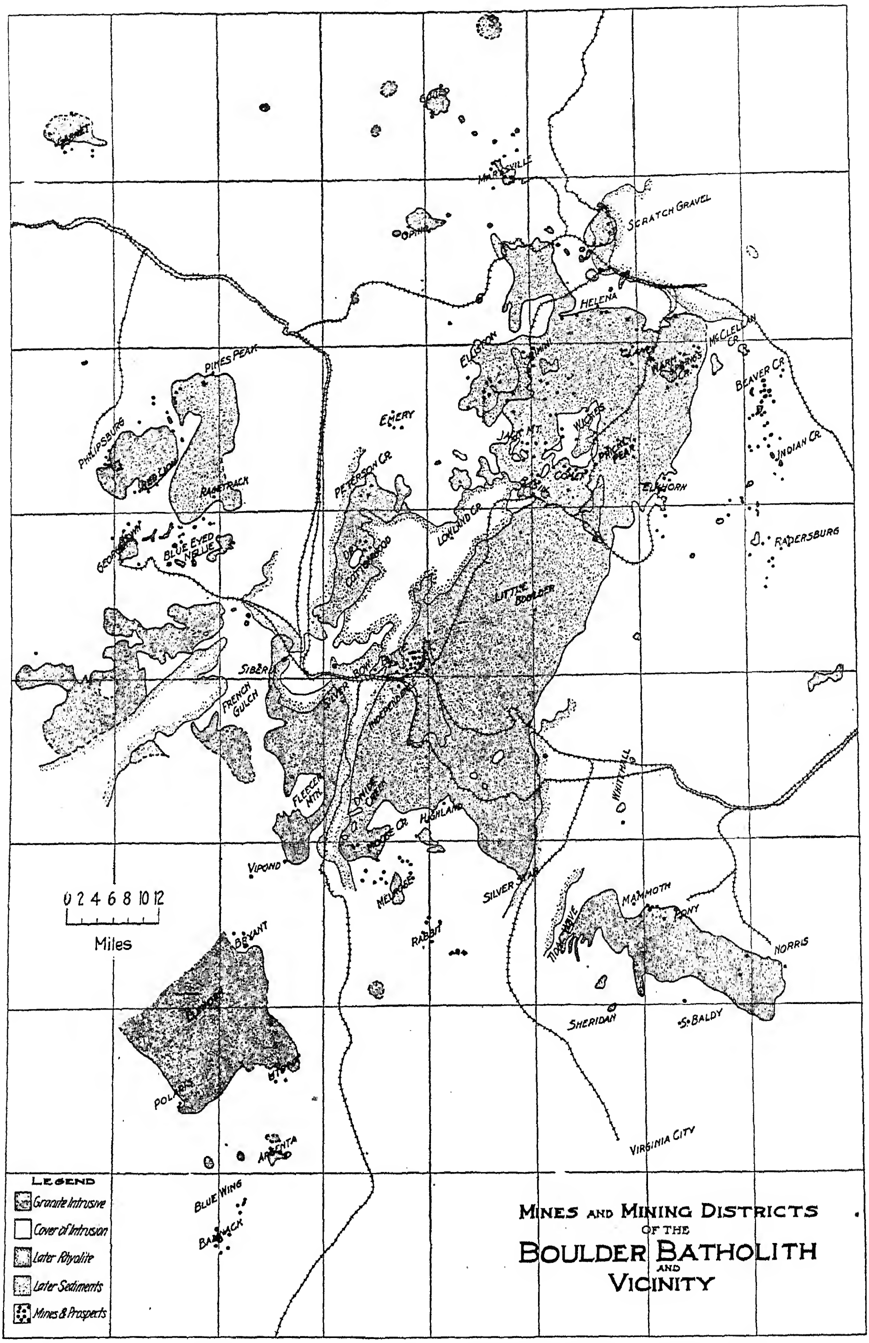



FIG. 1.—MAP OF GRANITE AREAS, MINING DISTRICTS AND MINES.

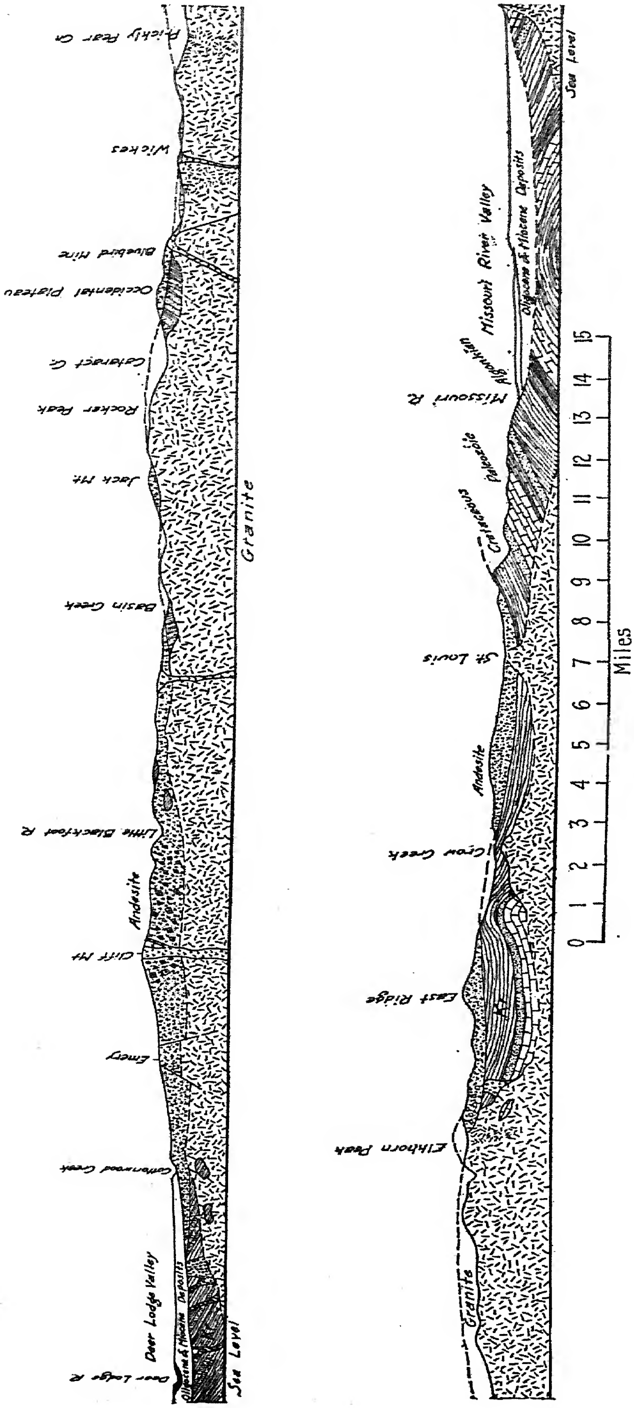



FIG 2.—TYPICAL SECTION ACROSS THE BOULDER BATHOLITH.

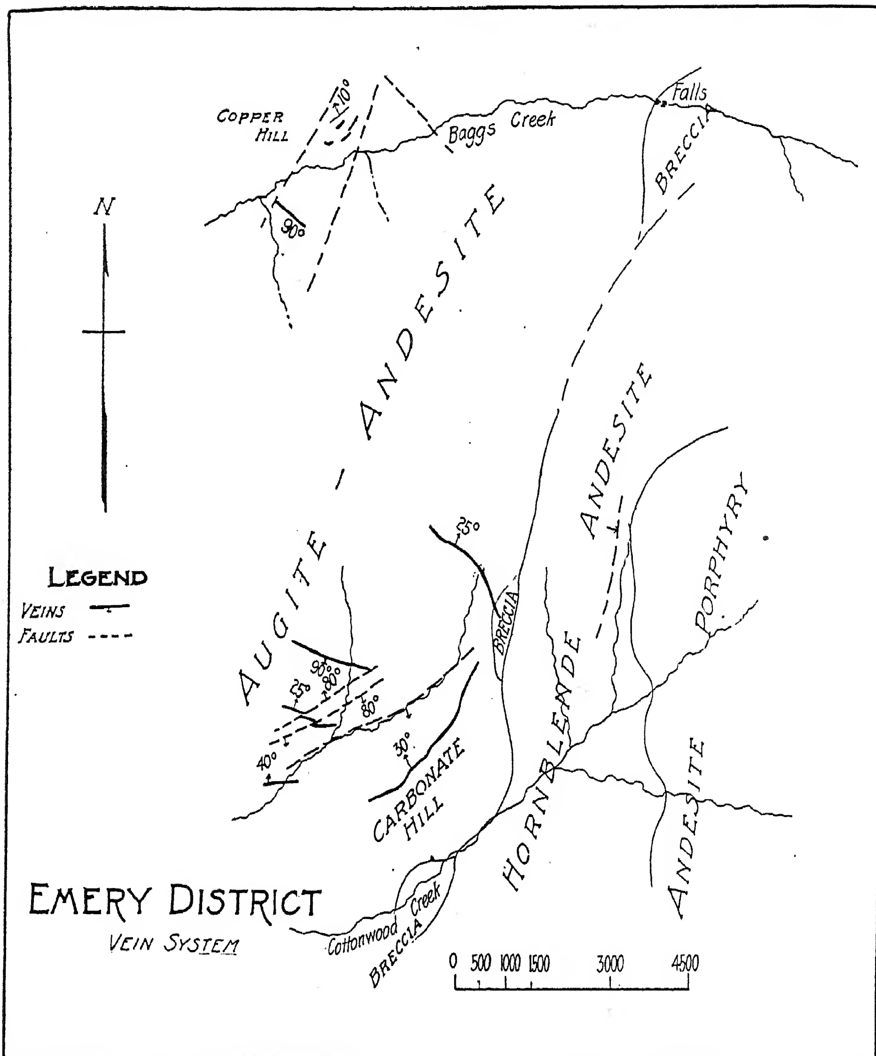



FIG. 3.—EMERY DISTRICT.

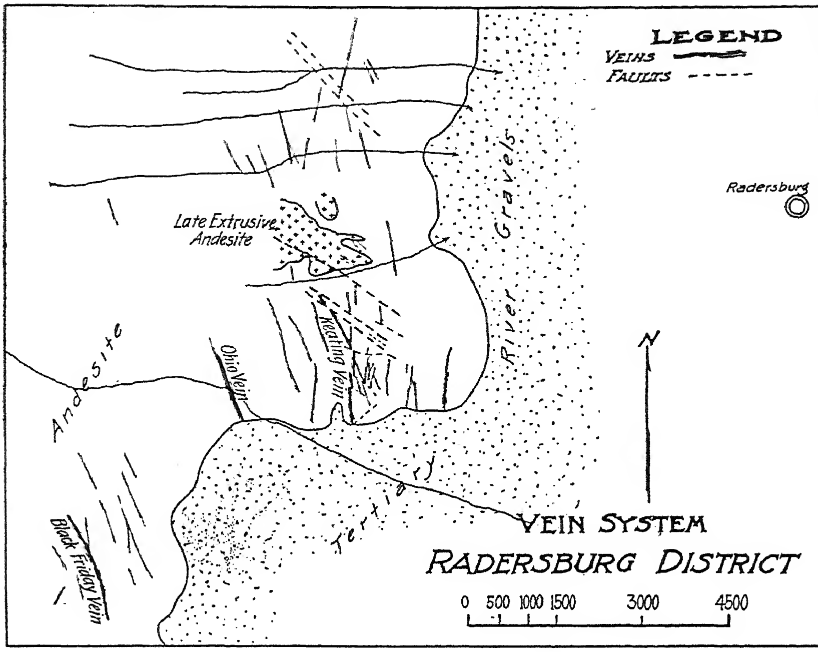



FIG. 4.—RADERSBURG DISTRICT.

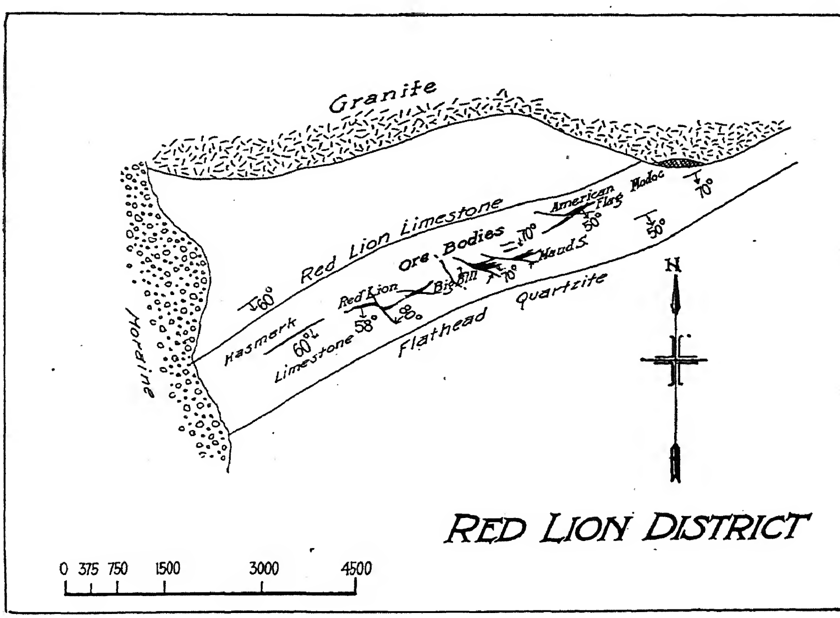



FIG. 5.

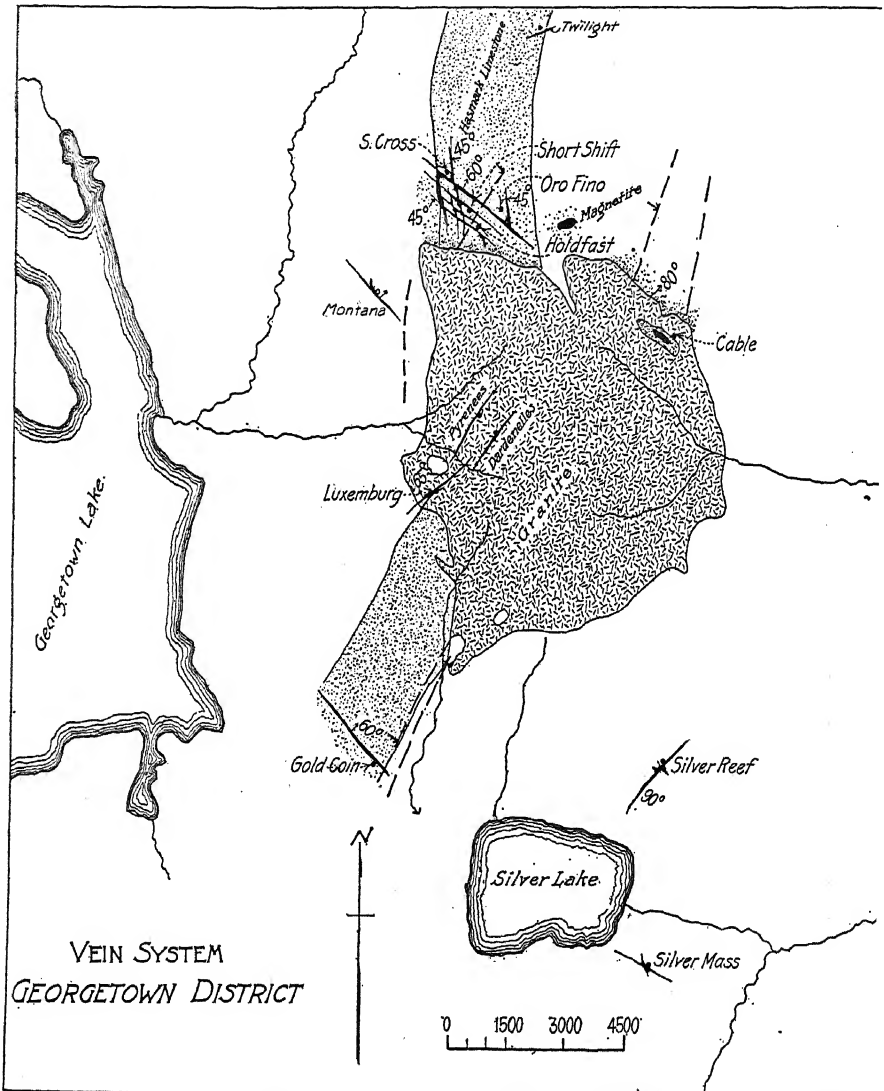



Fig. 6.—Vein system, Georgetown District.

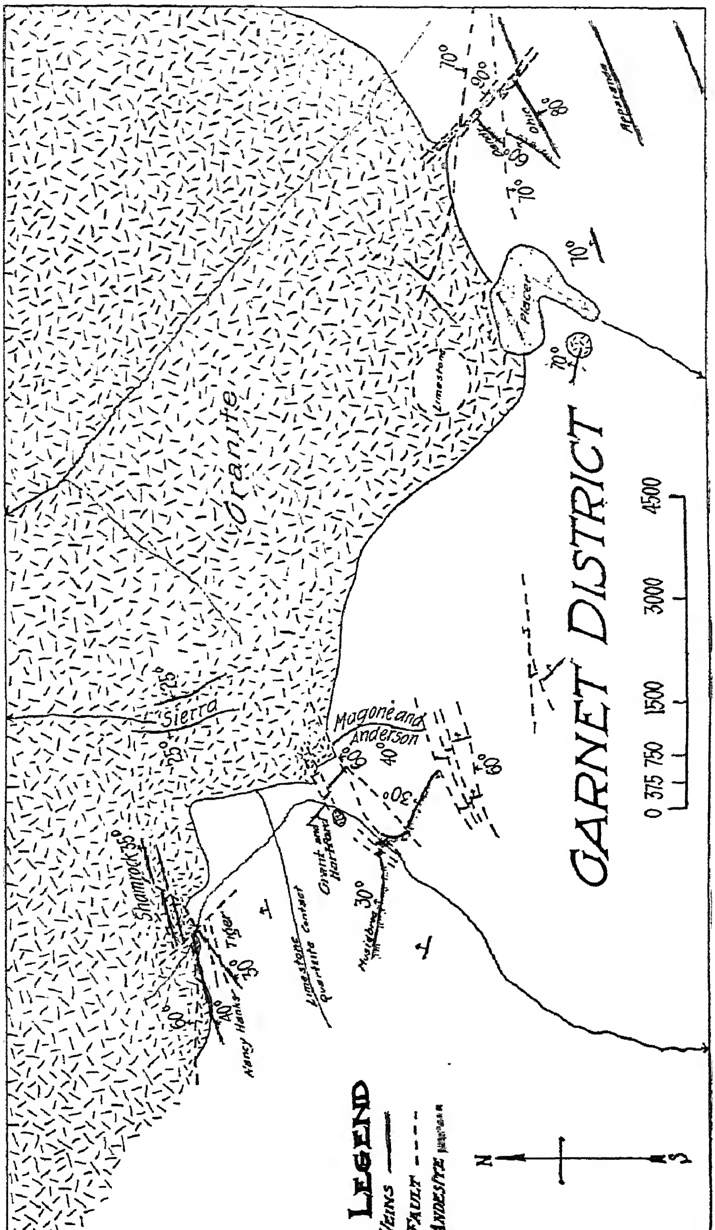

GARNET DISTRICT
Granite
Sierra
Magone and Anderson
Limestone
Placer
Shamrock
Tiger
Nancy Hanks
Grant and Hartford
Limestone Quartzite Contact
LEGEND
VEINS
FAULT
ANDESITE
0 375 750 1500 3000 4500
N
S

FIG. 7.

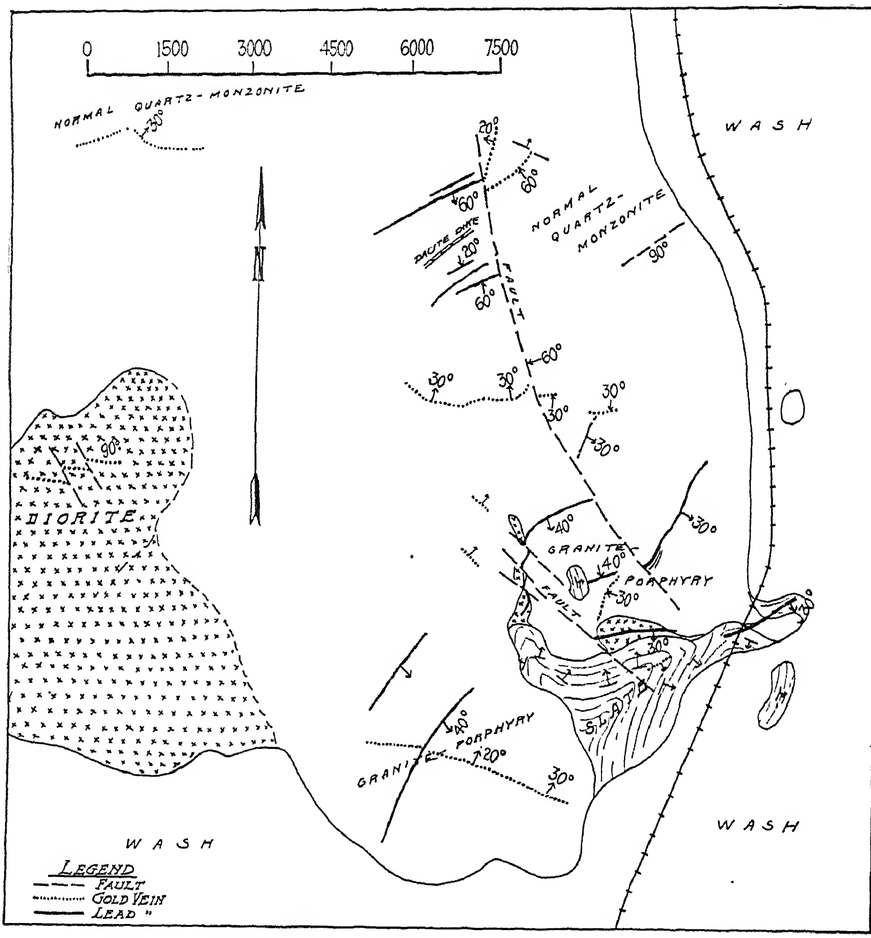



FIG. 8.—SCRATCH GRAVEL MINING DISTRICT, HELENA, MONT.

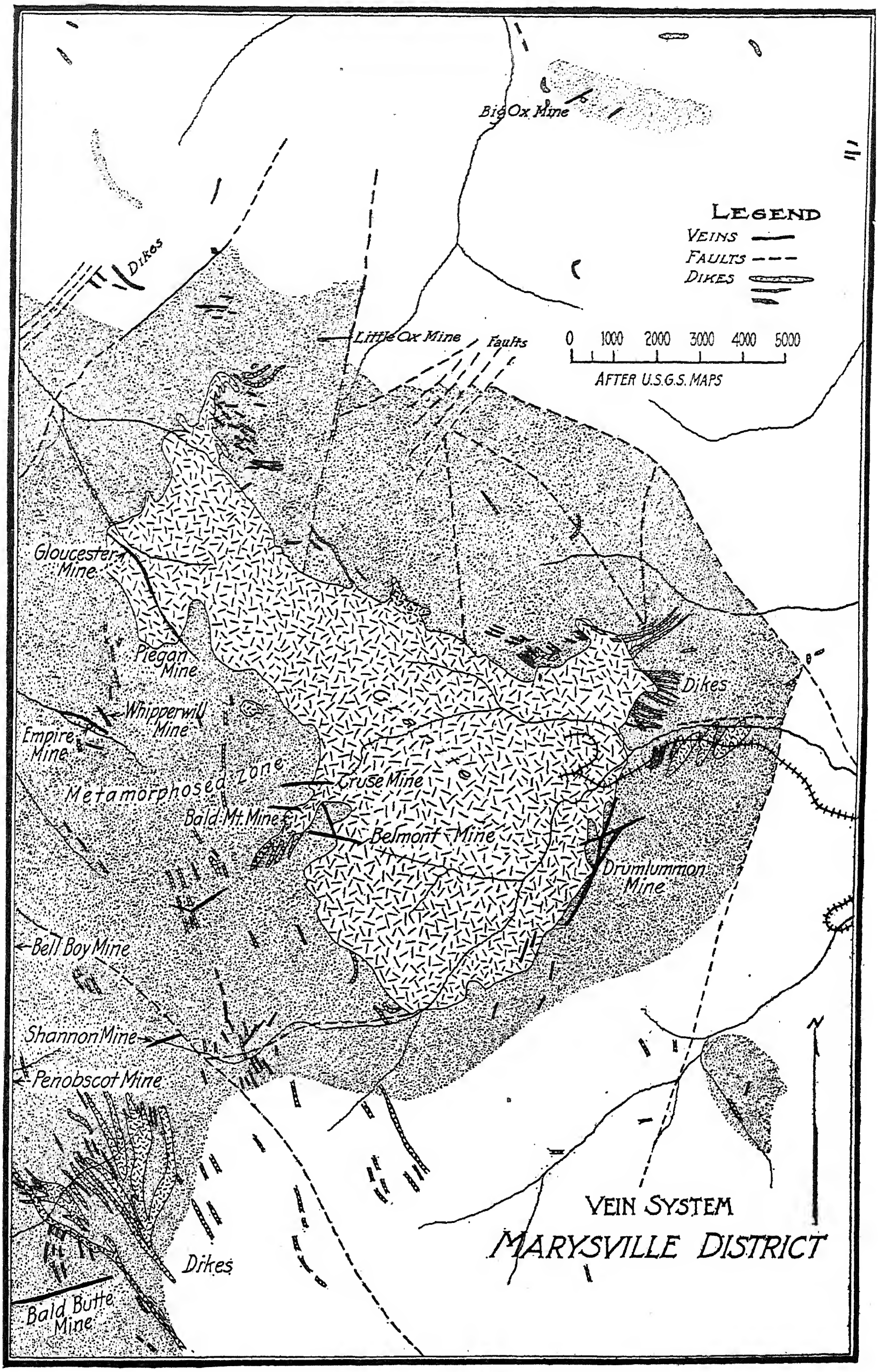
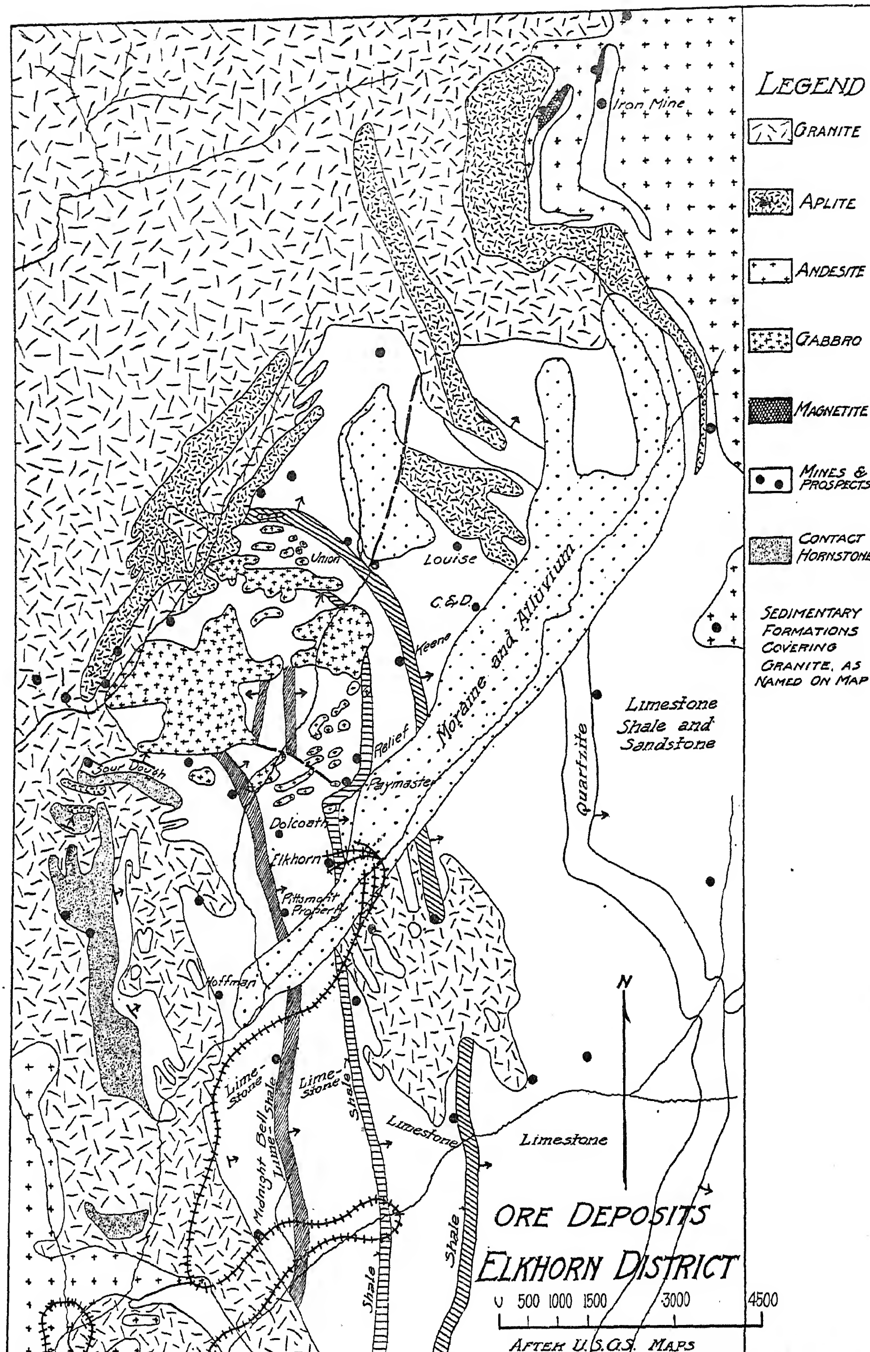



FIG. 9.—VEIN SYSTEM, MARYSVILLE DISTRICT.


LEGEND
GRANITE
APLITE
ANDESITE
GABBRO
MAGNETITE
MINES & PROSPECTS
CONTACT HORNSTONE
SEDIMENTARY FORMATIONS COVERING GRANITE, AS NAMED ON MAP
Iron Mine
Union
Louise
C. & D.
Keene
Moraine and Alluvium
Relief
Paymaster
Dolcoath
Elkhorn
Pittsmont Property
Hoffman
Sour Dough
Limestone Shale and Sandstone
Quartzite
Lime-stone
Lime-stone
Midnight Bell
Lime-Shale
Shale
Limestone
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N
ORE DEPOSITS
ELKHORN DISTRICT
0 500 1000 1500 3000 4500
AFTER U.S.G.S. MAPS

FIG. 10.

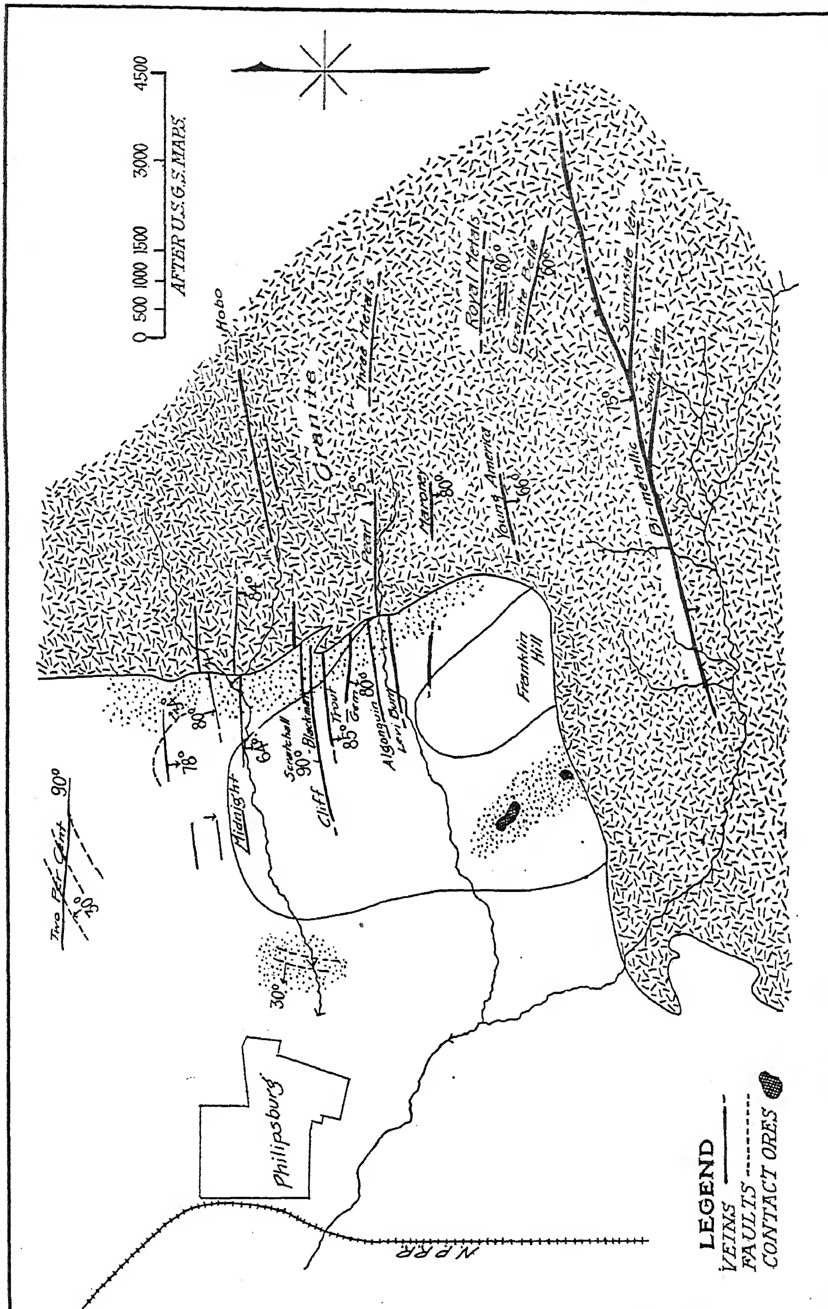



Fig. 11.—Vein system, Philipsburg District.

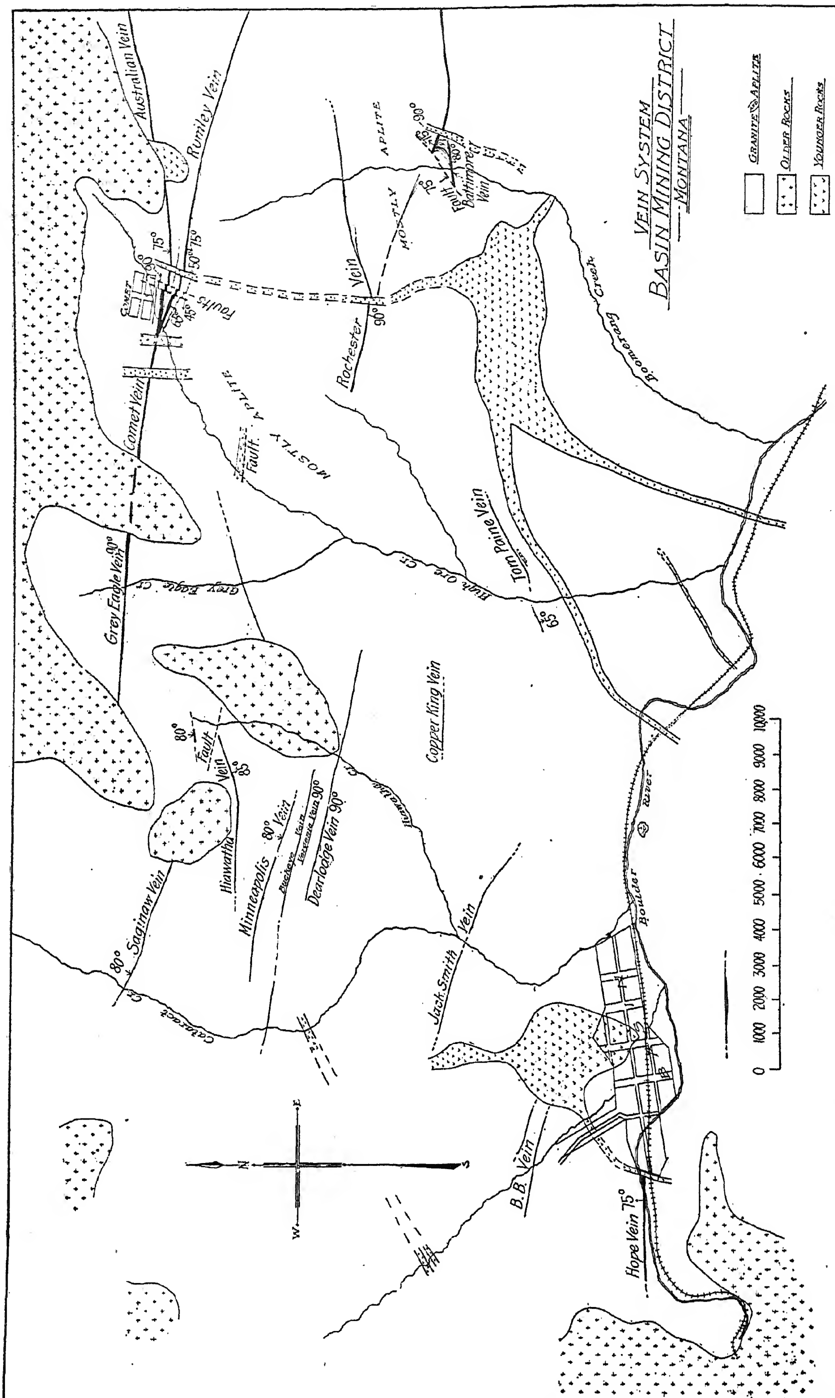



Fig. 12.—Vein system, Basin Mining District, Montana.

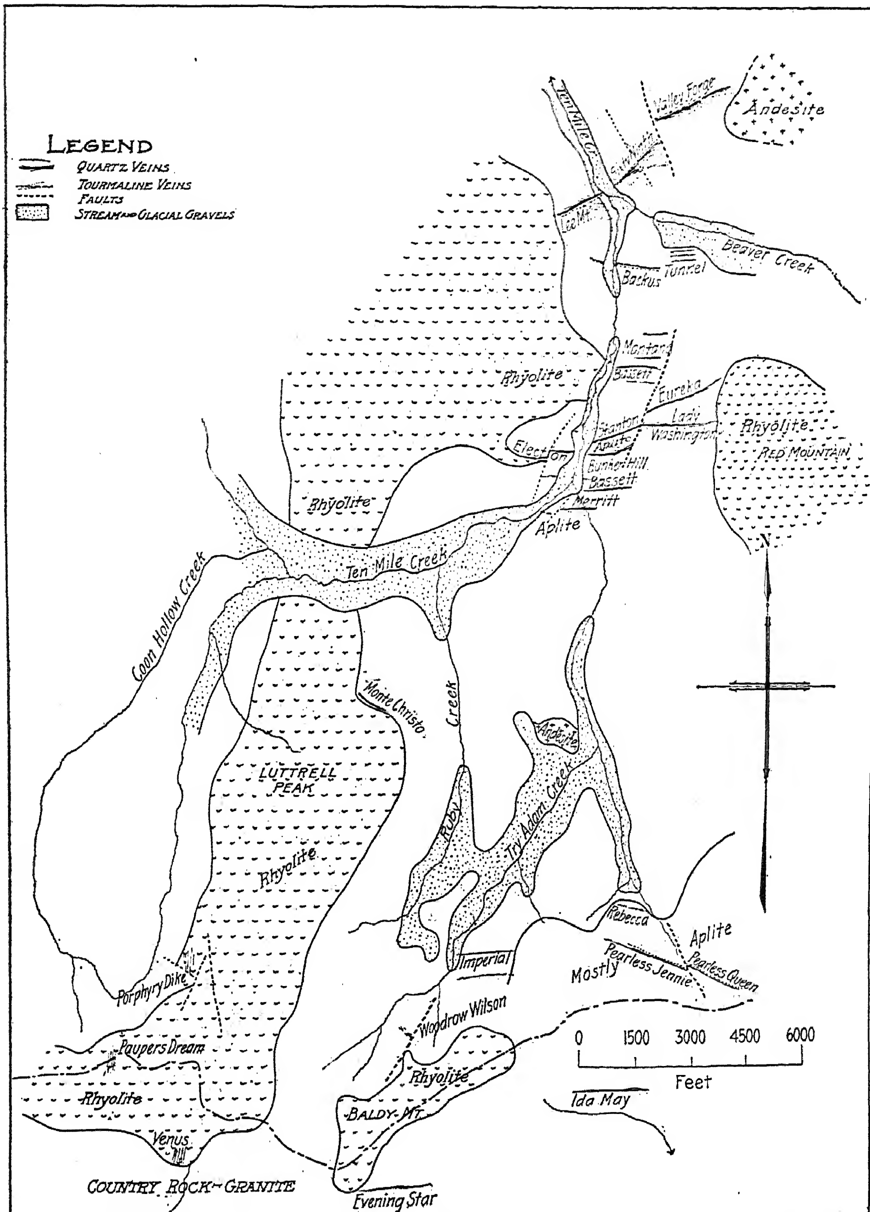



FIG. 13.—GENERAL MAP OF THE RIMINI DISTRICT, LEWIS AND CLARK COUNTY, MONTANA.

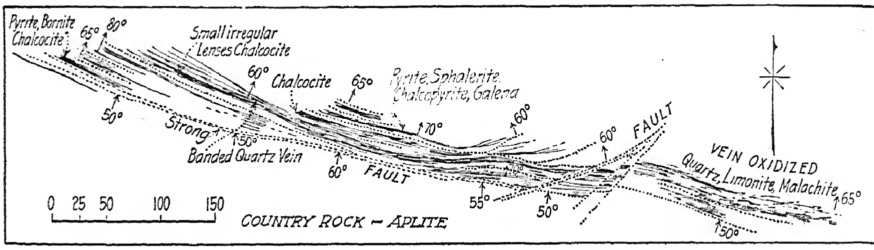



FIG. 14.—VEIN STRUCTURE, CRYSTAL MINE, JACK MT. DISTRICT.

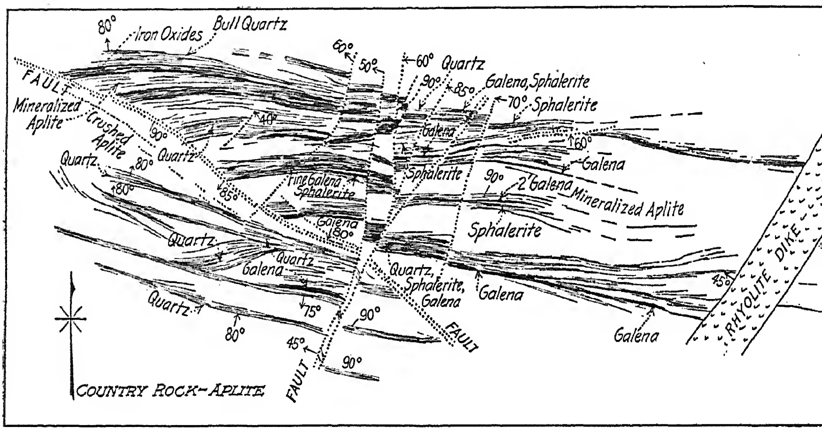



FIG. 15.—VEIN STRUCTURE, COMET MINE, BASIN DISTRICT.

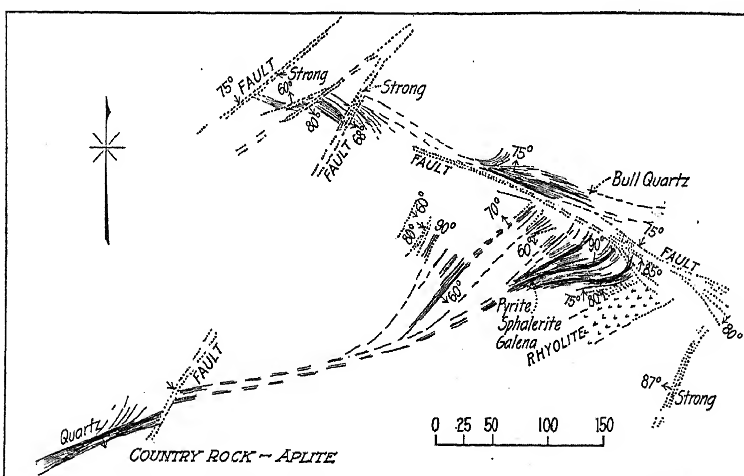



FIG. 16.—VEIN STRUCTURE, BALTIMORE MINE, BASIN DISTRICT.

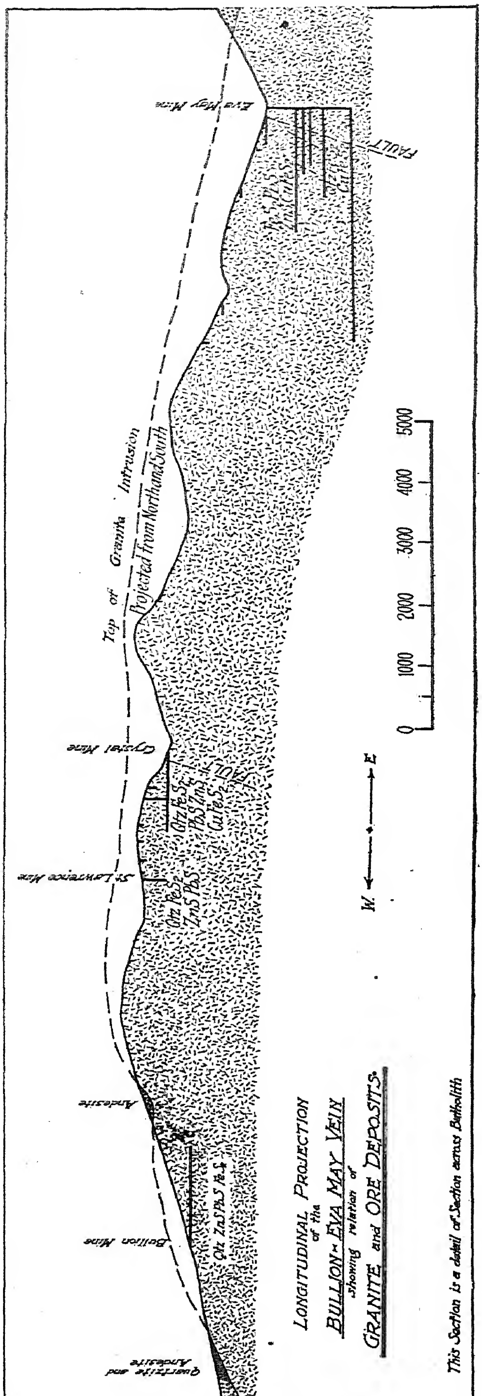



FIG. 17.—BULLION-EVA MAY VEIN WITH REFERENCE TO TOP OF GRANITE.

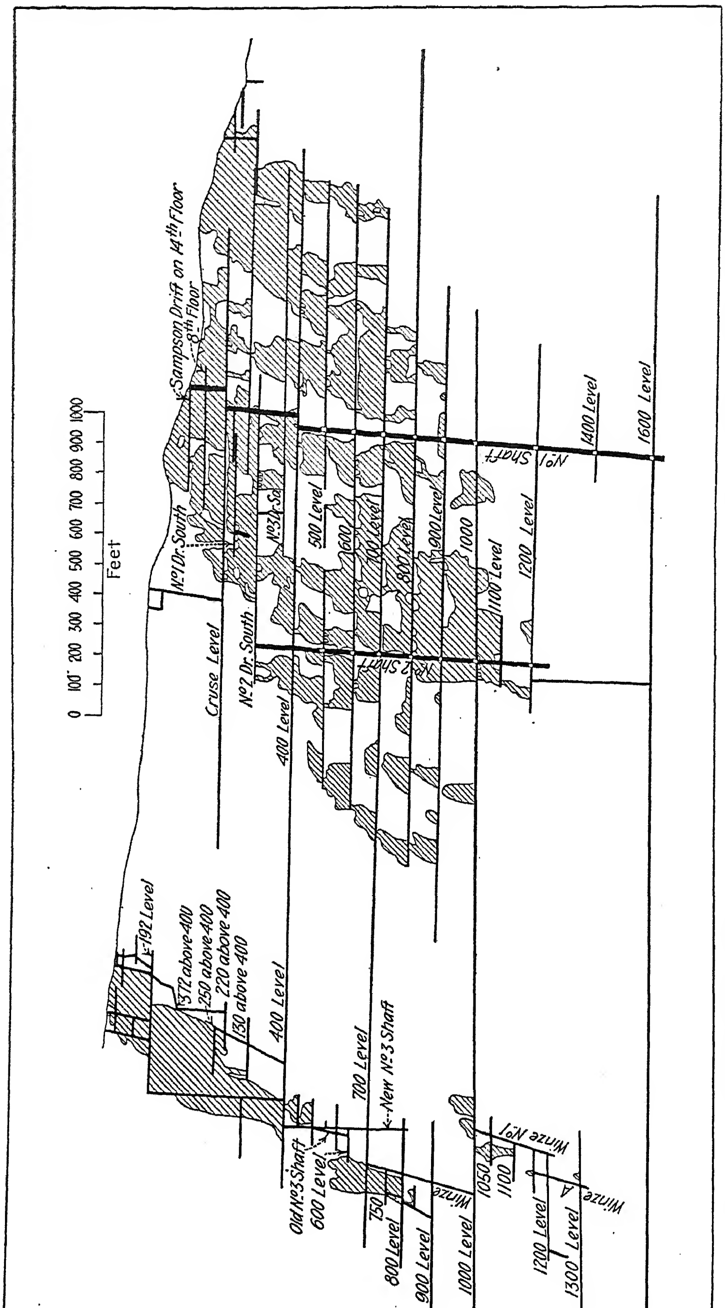



Fig. 18.—Drumlummon mine, longitudinal stope section. (After C. W. Goodale, *Trans.*, **49**, 269.)

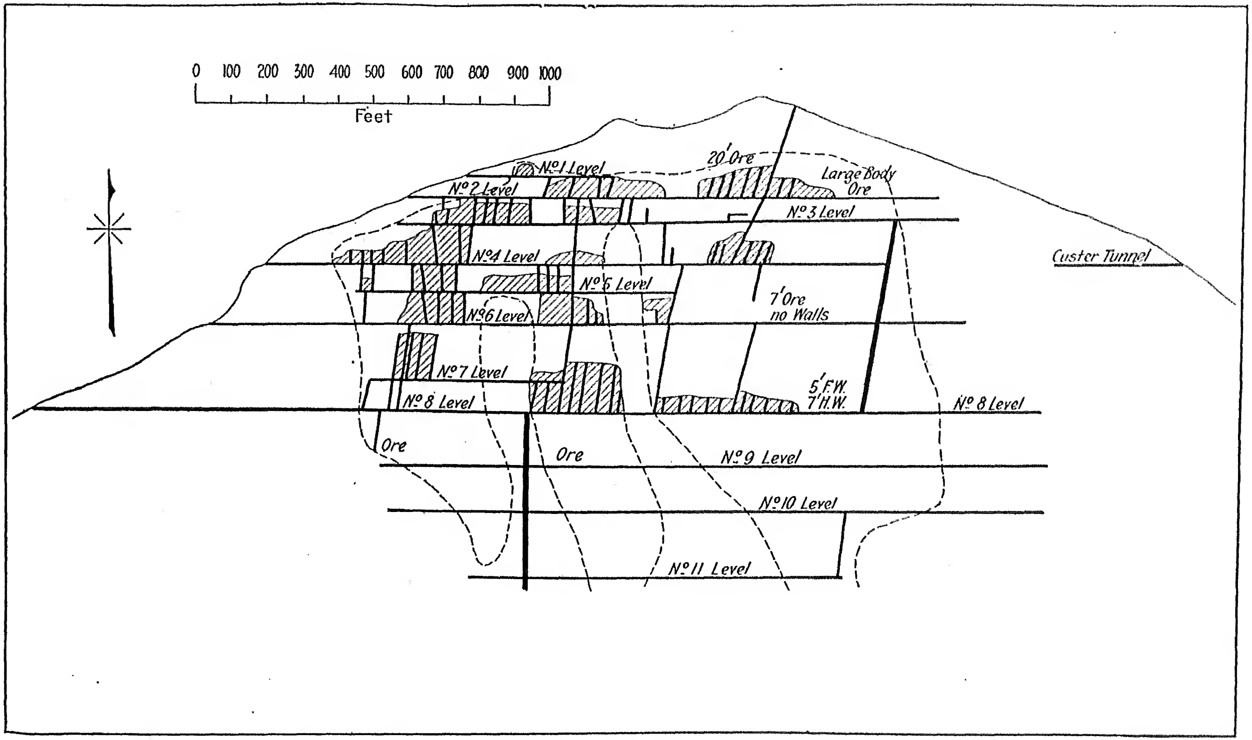



FIG. 19.—LONGITUDINAL SECTION, ALTA MINE, WICKES DISTRICT MINE.

(This section is projected on an E. W. plane. Dotted lines show boundaries of ore shoots. These ore shoots and the lower levels are taken from a map of Oct., 1893, shortly before the mine was closed. The stopes are from a map of April, 1888. In Oct., 1893, the mine was practically worked out above the 800 level, but ore remained between the 800 and 1100 levels. A total production of 997,650 tons is recorded, and about $2,500,000 was distributed in dividends. Later stope sections were destroyed by fire. The vein is developed 800 ft. below No. 8 level.)

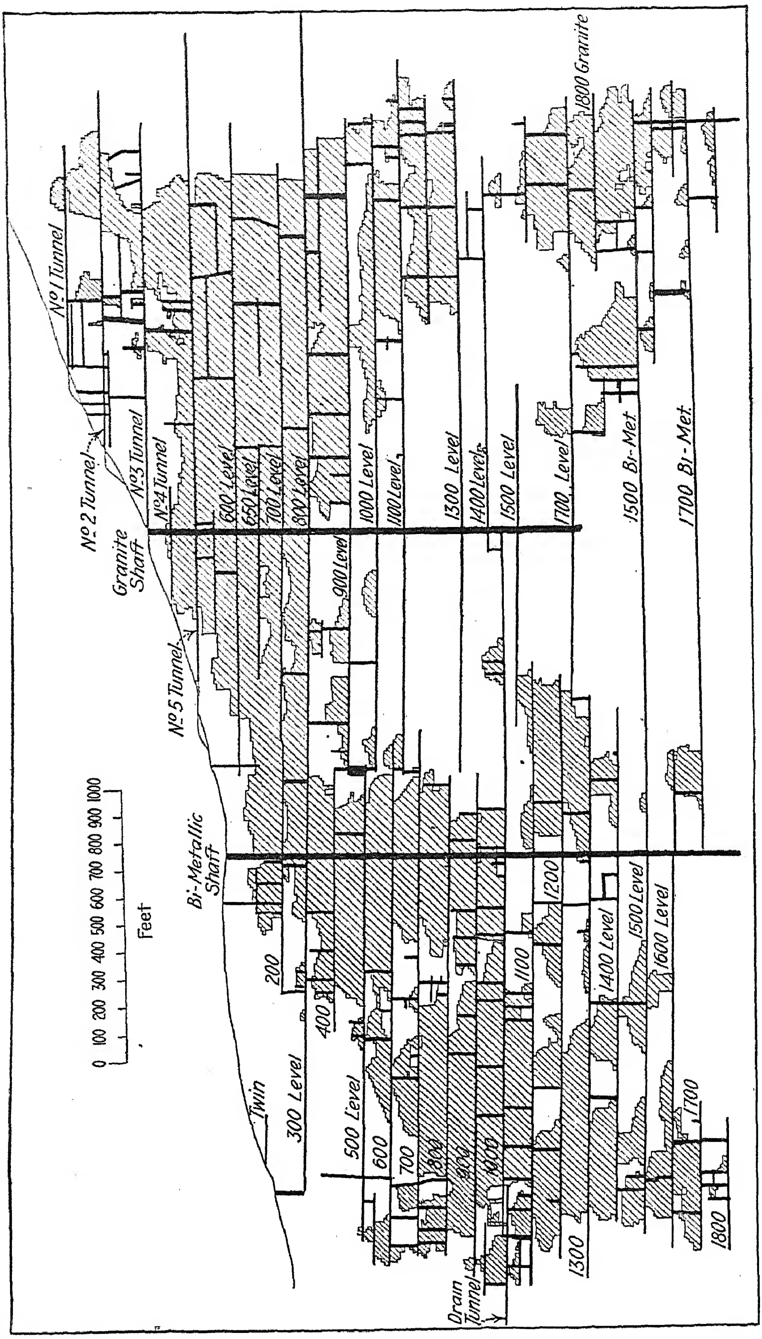



FIG. 20.—GRANITE-BI-METALLIC LONGITUDINAL STOPE SECTION.

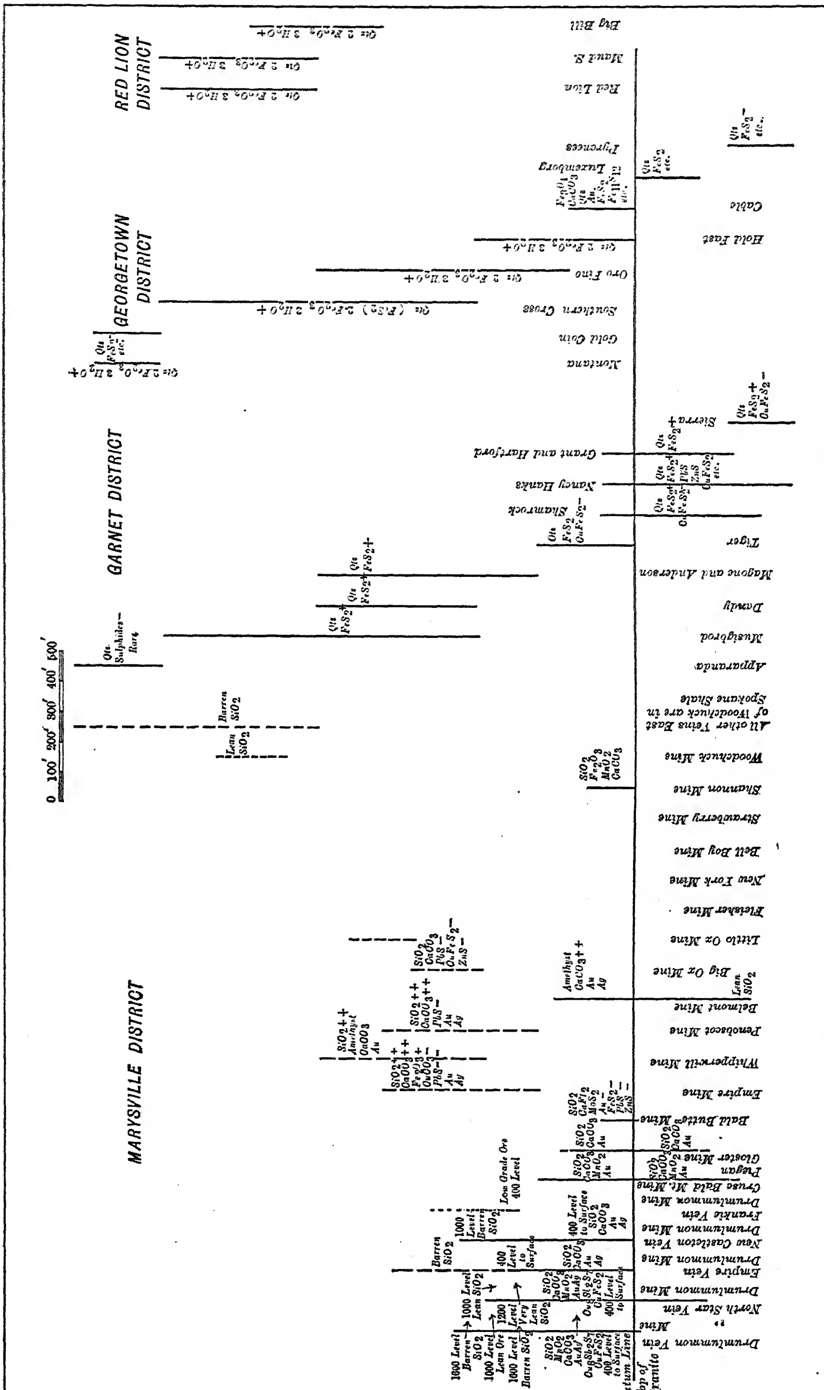



FIG. 21.—MINERAL DISTRIBUTION CHARTS OF GOLD DISTRICTS.

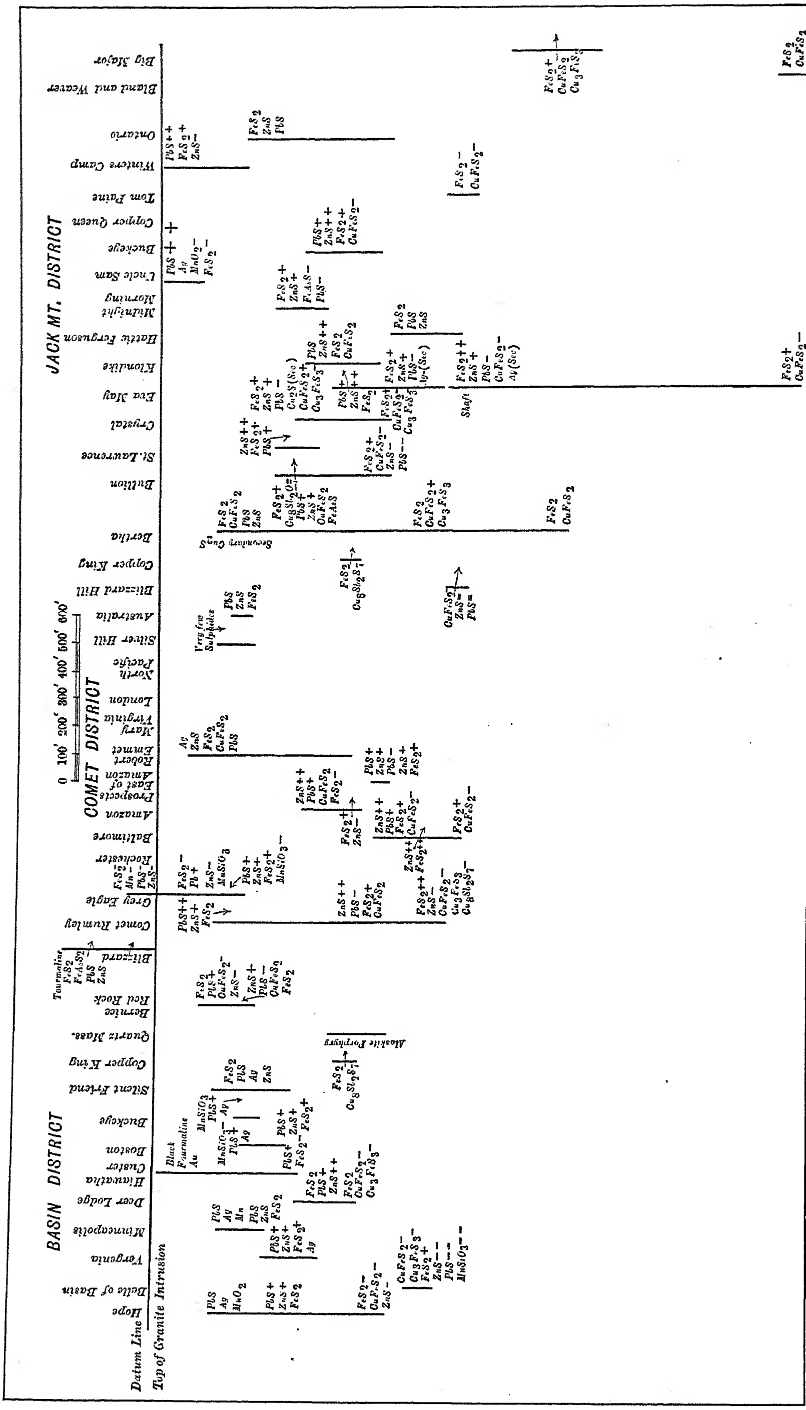



FIG. 22.—MINERAL DISTRIBUTION CHART, SILVER LEAD-ZINC DISTRICTS.

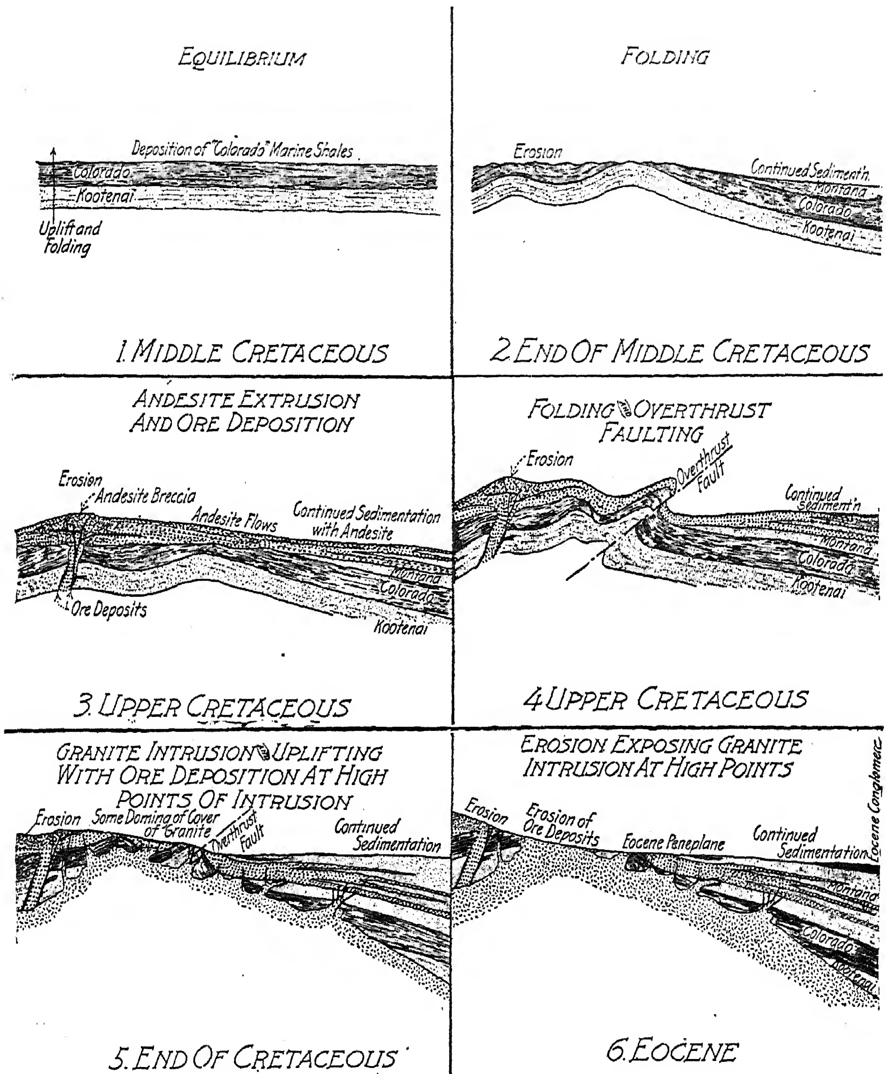



FIG. 23.—GRAPHIC REPRESENTATION OF GEOLOGIC HISTORY UP TO GRANITE STAGE ORE DEPOSITION.

(Idealized to show age relationship of igneous rocks and cretaceous period. Most of the ore deposits are in rocks of other ages.)

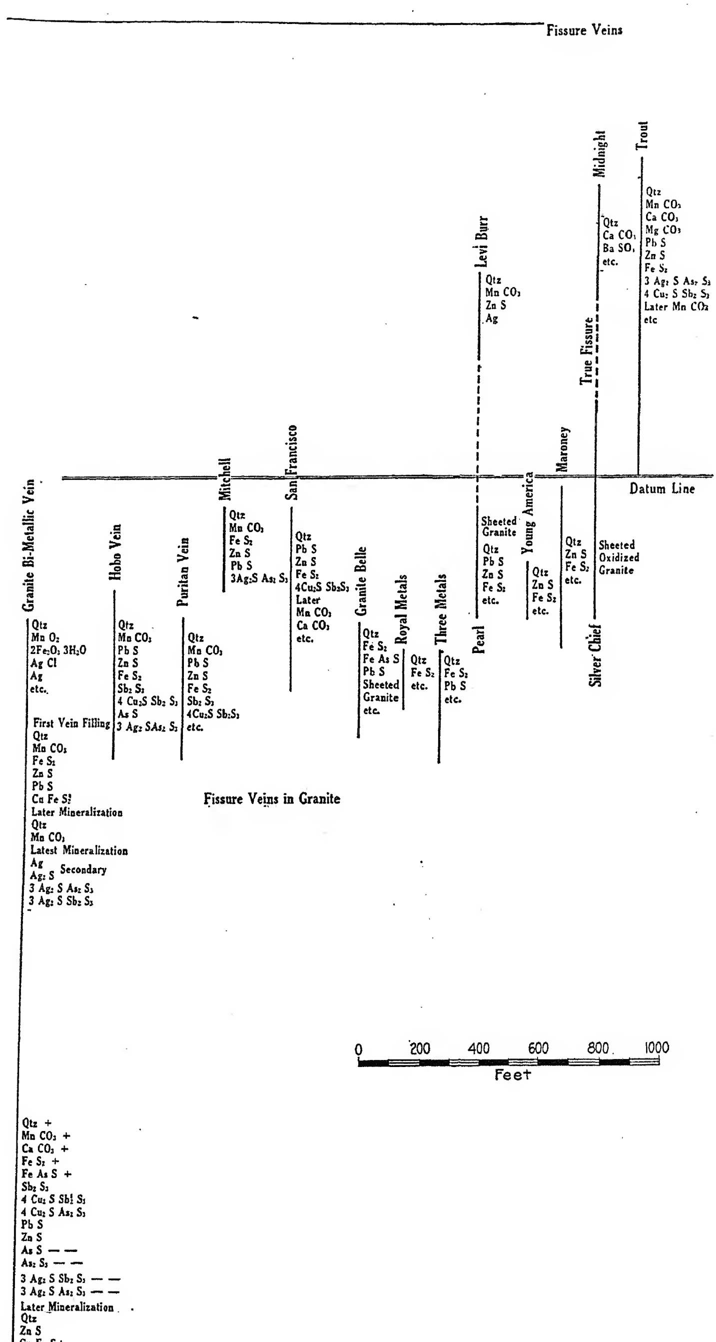



FIG. 24.—MINERAL DISTRIBUTION

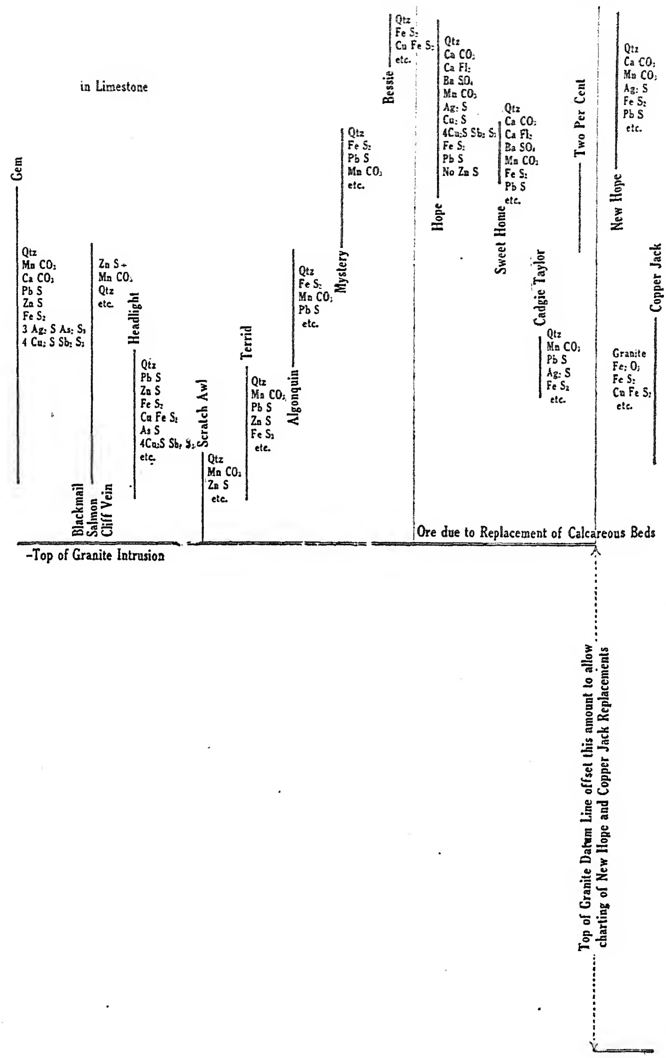



CHART OF PHILIPSBURG DISTRICT.

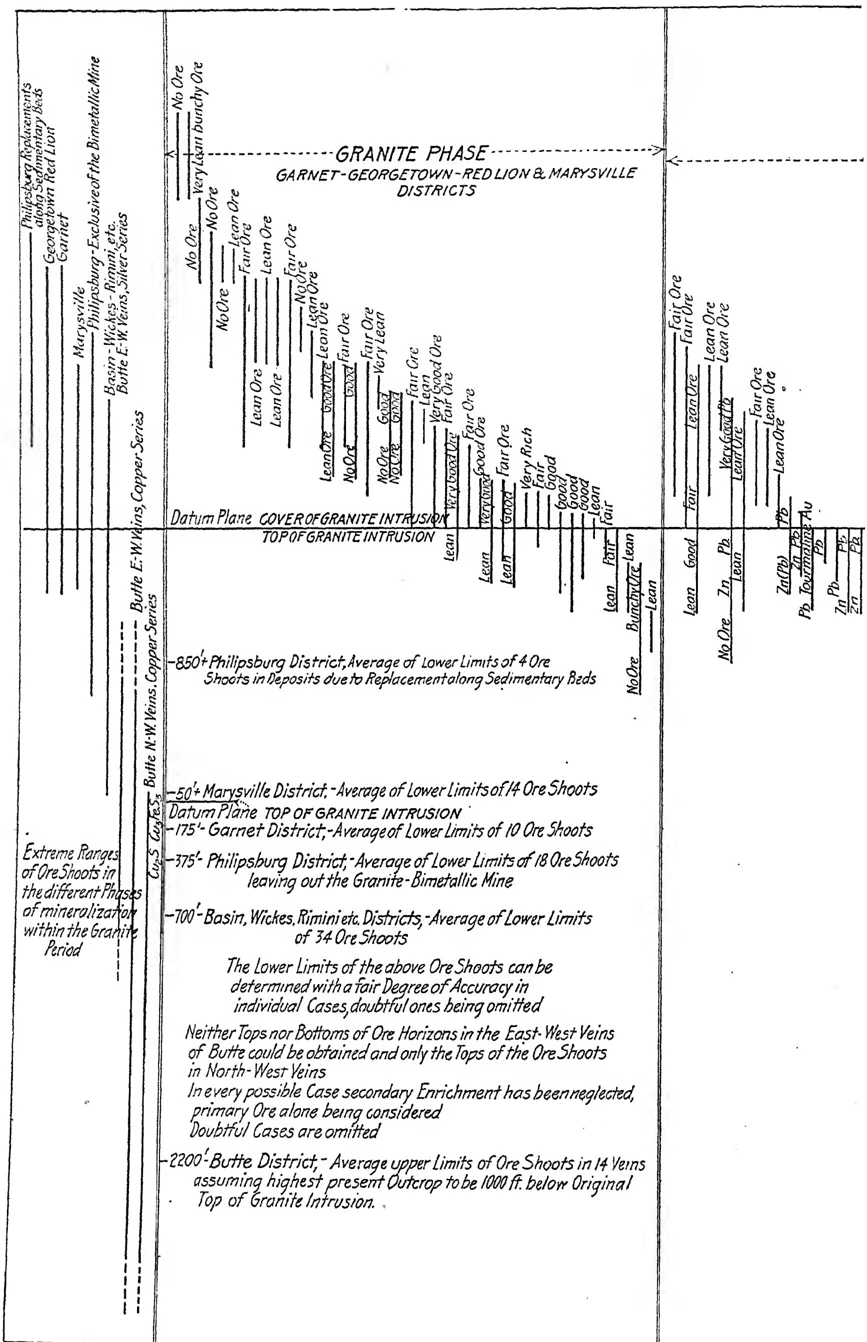
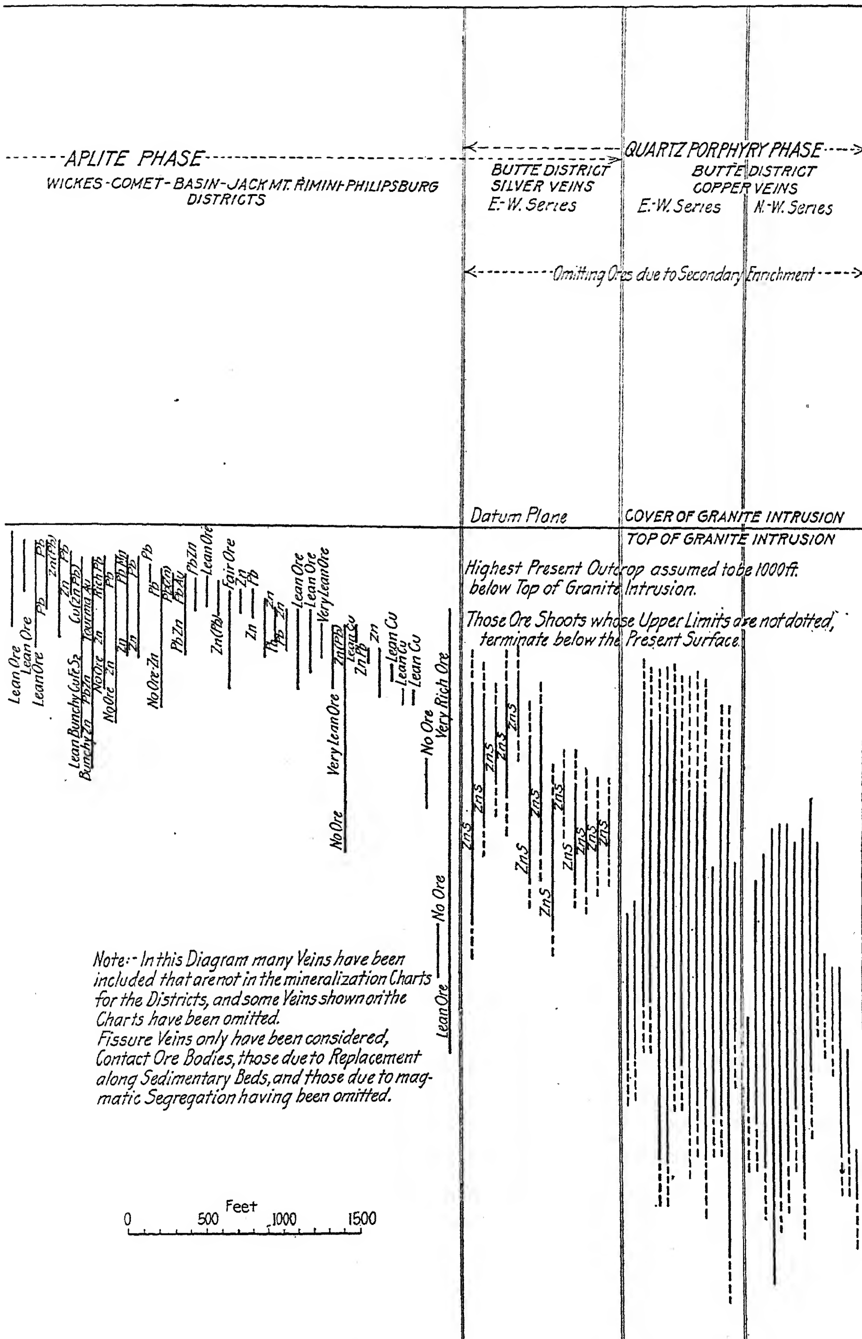



FIG. 25.—DISTRIBUTION OF ORE SHOOTS IN FIS-
(See R. H. Sales, *Trans.*, **46**, 3,




SURE VEINS, GRANITE PERIOD OF VEIN FORMATION.
for mineralization of Butte veins.)

## Discussion

Walter E. Gaby, Vanadium, Colo. (with discussion*).—Having enjoyed associations with the authors of this paper while on the Anaconda geological staff, I want to express my appreciation of it as a contribution to economic geology very direct in its application to ore deposits. The conclusions reached seem to me to show that concrete generalizations can not be made over very wide areas as to structural relations of ore deposits, but taking each district separately data of great value has been presented. Looking over thin sections of Montana rocks, among these a collection belonging to Mr. Billingsley and some prepared by myself, one important point in the Butte area has come to my attention which I would like to discuss.

Related to the batholithic intrusion, there are a number of various types of both the earlier andesites and the later rhyolites, especially of the rhyolite. Some of this rhyolite is composed so predominantly of plagioclase and quartz as to be more properly called dacite, and near the volcanic centers the different phases are even more numerous, the quartz in some becoming a very minor constituent and confined to the groundmass. Some portions of these rhyolitic extrusives assume deep reds and grays, and in the hand specimen may very easily be mistaken for andesite of the older period that has been altered by weathering or by baking near the intrusive granite contact; but thin sections of such specimens remove this color effect and present so clearly all the minute details of texture and mineral composition that they are in every respect identical and indistinguishable from the commoner types of the rhyolite. The andesites generally are augitic rocks with smaller plagioclase feldspars than the rhyolites and exhibiting a different arrangement. Therefore the so-called red and dark gray andesite near Butte is not a portion of the pre-batholithic rocks, but a phase of the rhyolite, and the authors, in assuming that "roof pendants or large inclusions of andesite are found within the productive area itself" (note 45), err in this particular. My conclusion is borne out by the early observations of Emmons and Tower, who specifically state that no andesite occurs within many miles of the Butte district.

This has some bearing on the authors' datum line, the top of the granite intrusion, affecting conclusions in this area as to the position in the fissures of ore shoots related to it; and I am inclined to believe that over a portion at least of the Butte area this reconstructed contact surface occupied a higher position than they give it, hence increasing the relative depth at which the different ores were formed here.

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* Received Aug. 20, 1917.

JOHN B. HASTINGS, Los Angeles, Cal. (written discussion*).—This paper is so admirable for its brevity that one hesitates to enter into discussion. The cycle of eruptives makes one wonder, "Where's the basalt?" and the youthfulness of the Idaho flows suggests they may yet come.

*Veins and Dikes.*—Veins and dikes seem to be closely aligned in the minds of some authors, while to me they appear quite distinct, the first representing quiet, long-continued action, typified by metasomatic replacement, far removed from volcanic action; the latter, sudden and cyclonic, emanating from the heart of vulcanism. The rock openings occupied by both types were formed by kindred forces, but in the one they were closed for indefinite periods before invasion by the solfataric solutions, and in the other their formation and occupation were simultaneous.

*Open Fissures.*—I do not believe in the existence of open spaces of discission that may be occupied by veins. Occasionally they occur; I found one with a machine drill 350 ft. below surface, fortunately in an adit; the water spouted for 12 ft. in a 6-in. stream for nearly 24 hr. It was only a flattish fissure, 6 in. wide, crossing the perpendicular vein, and indicating how tabular veins of ore might be formed. The practical absence of open fissures should be a joy to miners. We all know the misgivings with which we feel about for an old winze having only 30 ft. of water in it, in an unsurveyed and inaccessible part of a mine.

It is said that veins are sometimes formed in open spaces left by protuberances of the walls, which meet after having been displaced; but I believe that in such circumstances there would be sufficient attrition and brecciated material to fill the fissure, though perhaps loosely. Parts of veins may possibly have been formed in originally open spaces, but if so, the spaces were made by solution shortly preceding the deposition of the veins. That total solution of rock adjacent to a fissure may occur under favorable conditions is proved by the existence of large bodies of sulphide which contain only a few per cent. of silica; while on the same level, and in other levels above and below, all being within the same ore shoot, assays may show as much as 50 to 60 per cent. of silica, although the gold, silver and copper assays may be almost identical in all the places. Open spaces of solution are common, but some Providence keeps them dry. We find such spaces in the Santa Rosa mine, near Mazapil, Mex., partly filled with ore and detritus and extending 1000 ft. deep.

*Contraction Cracks.*—I have not observed contraction cracks in dikes, due to cooling, which would be conducive to vein formation. Where I have encountered them, the dikes have been denser than the enclosing country rock, except when shattered by movements after cooling.

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* Received Oct. 8, 1917.

Where such later movement has not occurred, I think that solutions starting to ascend the dikes would soon be deflected into the walls. Cooling cracks observed in large dikes have been normal to the walls, simulating the columnar structure of basalt.

Regarding the fissures filled by aplite veins (p. 307), I take it that they are held to be the direct result of contraction of the batholith and are parallel to its shorter (east and west) axis, and also occur ordinarily in reopened aplite dikes. My idea of the cooling effect of a homogeneous magma would be a checker-board jointing and a settling of the mass, causing normal faulting near the east and west boundaries, parallel with the longer (north and south) axis. I should think that the aplite dikes followed by the veins indicate that a stress had already occurred against these particular points; their reopening or vein fissuring denotes that the stress continued, while the longitudinal post-mineral faulting showed that it remained active.

Without considering whether the mineral content of veins was leached from the magma or carried up through it by primeval waters, it can be understood how the final cooling and crystallization caught within it waters which might leave a residual mineral content. Some of the extruded aplites, as described in the paper, contain a primary mineralization. It would be interesting to know the age of these sulphides and their value. This might also be said of pyrite in the granite at Red Rock Creek, and, indeed, of any sulphide that occurs as a primary mineral in these igneous rocks, enriching them to a commercial value, or nearly so.

In 1885, at the Pioneer mine, Boise Basin, Idaho, an andesitic dike, 30 ft. wide, was being stoped for 200 ft. in length by 300 to 400 ft. deep, yielding $7 gold per ton. In examination, much hinged on whether the mineralization was primary. The dike was irregularly and slightly fractured and its freshest parts contained sulphides which I panned and assayed; they may have carried a trace of gold, although the seams were undoubtedly rich, as free gold was common in them. In the whole stope, in these seams I could find only two pieces of quartz thick enough to be detached, about 2 in. square, ½ in. thick, with no visible gold. They assayed $2300 and $2700 per ton. I concluded that a vein fissure of the district had crossed the dike diagonally, causing the fracturing, and deposited one of the Boise Basin rich shoots, which, if formed in an ordinary fissure, would have consisted of white quartz, pyrite, and free gold, such as adorned the cabinets of the district.

*Vertical Distribution of Ores.*—Without doubting the zonal deposition of lead and zinc minerals, which seems harmonious, it would be weighty evidence if the mine managers of the Coeur d'Alene would tell what has occurred there. They carry a mental map of each level, showing the areas stoped as cleaner lead ore and those avoided as zincky. In

the one, prospecting was perhaps vigorous; in the other, slack. With the higher price of zinc, such discrimination would not be seen in recent working levels.

It was said that in the Coeur d'Alene the ore shoots did not outcrop, but I could not find this to be exactly the case except perhaps at the Hercules. The shoots were very short at the surface, lengthening in depth. The regularity of the fissuring and homogeneity of vein filling suggested secondary concentration downward along the ends of the shoots, but I have never heard this explanation. A possible reason for the lengthening of the ore shoots with depth is that the rising ore-bearing solutions were dammed back by the deposition, and were crowded laterally under the part of the vein already plugged. Assuming the difference in height of deposition of lead and zinc as only a few hundred feet, in an ore shoot 1500 to 2000 ft. deep there would be a long descending and continuous overlapping.

When Dr. Lindgren visited Wood River, the district was rather quiet. In the few properties being worked, the evidence showed an increase of zinc with depth; perhaps the older mines, had they lasted, would have shown the same. The only large active mine at that time was the Minnie Moore. Others which worked in the early 80's, and were mainly responsible for the Wood River boom, were: The Bullion, the Mayflower, and the Jay Gould, producing about $3,600,000 from galena, with some carbonate, averaging 20 oz. silver to the unit of lead; at 300 to 400 ft. depth the vein was faulted, and has not been found, but the ore had not changed in character. The Idahoan, near by, produced about $500,000 from galena, lower grade in silver, and there had been no change in the ore when the mine stopped. The Red Elephant and the Red Cloud, each with an output of about $1,000,000 from high-grade silver-lead ore, terminated without change of ore; the latter mine was supposed to be faulted, but it had also pinched to a few inches. The foregoing mines are in the Bullion area, together with Falk's Star mine, near Hailey, which had the same general history.

At Ketchum, the Elkhorn produced $1,200,000 from 300 ft. in depth, where the vein faulted and is not known to have been recovered; the ore was galena, with some carbonate, and averaged 2 oz. silver to the unit of lead, a trifle more in the bottom. The Baltimore and Independence, on the same vein system, produced perhaps $300,000 of lower-grade lead-silver ore; there was no change in the ore. The North Star, on the east end of the same vein system, was heavy in zinc from the surface, and careful assay maps would be needed to prove whether that metal has increased. The Parker, in the same district, produced nearly $300,000 from very high-grade lead-silver ore and was quickly bottomed; possibly the rather obscure vein was faulted. At Boulder, the Trapper, Ophir, and Yankee Girl vein yielded galena high in silver and gold, the product

being perhaps $150,000. The Trapper was rather zincky from the surface, but the orebodies terminated without change in character. At Smokey, the Silver Star, Carrie Leonard, Isabella vein was also zincky; the oxidized ore rich in concentrated silver, from which the zinc had been dissipated, was limited, and the mines made little geological history. The King of the West was a small, well-defined lead vein which paid for a slight depth; there was no change in the ore, but the yield had not been great enough to warrant much prospecting. On Boyle Mountains, the Ontario and Blackhorse were worked to a shallow depth, producing, say, $250,000. The veins are badly faulted, but there was no change in the ore. In this vicinity is the former Queen of the Mountains, now worked as a zinc mine. The vein contained on the surface about 3000 tons of 65 per cent. lead ore, with 180 oz. silver to the ton, but all the rest of the vein was zinc. The lead veins are faulted by the ordinary Wood River andesite in the Senate mine at Galena, and in the Bullion mine, at Bullion, but by great quartz prophyry dikes at Boulder and on Little Lost River.

MESSRS. BILLINGSLEY AND GRIMES (authors' reply to discussion*).—Since the publication of our paper, several things have come to our attention which have a close connection with the subject discussed, and these we shall dispose of before answering the published discussion.

*Acknowledgments.*—A very able paper[1] by B. S. Butler had escaped our attention until after the publication of our own effort. On page 119 of Mr. Butler's paper is a diagram which expresses exactly our ideas of the relation of veins to the form of intrusive, and also to its cooling and differentiation. This section may be compared with Fig. 23 of our paper, when many similarities will be noted.

H. G. Ferguson and A. M. Bateman had previously written on "Geologic Features of Tin Deposits"[2] and emphasized this relation of the top of the granitic intrusives to vein-forming activities. Doubtless, others have been impressed with the same phenomena.

The advance made in our paper is therefore confined to the definite correlation of the granite cupolas and their vein clusters with the main batholith, with its troughs between the cupolas, practically barren of mineralization. This association of vein-forming activities with the high points of intrusion was found to be true of every period of vein formation—granite, aplite, and quartz porphyry— and this was a further application of the theory than had previously been made. It may be regarded as excellent proof of the soundness of this observation that it

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* Received Feb. 12, 1918

[1] Relation of Ore Deposits to Different Types of Intrusive Bodies. *Economic Geology* (1915), **10**, 101.

[2] *Economic Geology* (1912), **7**, 223.

was independently made at such widely separated localities and with different types of ores and veins.

In studying a large area of Gilpin and adjacent counties in Colorado, Edson S. Bastin[3] and James Hill arrived at conclusions very similar to some of those which we have advocated for the Boulder batholith. These points of similarity are especially marked; first, in assigning the derivation of the commercial ore largely to solutions escaping from later porphyries during cooling; and secondly, in the regional and sequential variations noted in the composition of vein-forming agents.

*Arsenopyrite.*—An error which crept into our compilations of data is found in the vertical range diagram on page 329, where arsenopyrite is classed with the higher horizon, cooler temperature minerals. This is not the case, as arsenopyrite is a higher temperature residual mineral from earlier vein-forming solutions, and is so described on page 314 of the paper.

*Additional Notes on Butte.*—We have lately considered it advisable to add to the original discussion some notes on Butte, which follow:

Aplite may be seen grading into quartz porphyry in several places. Crosscuts run under large surface outcrops of quartz, or of quartz with sulphides or oxides of ore minerals, have shown the absolute lack of mineralization at moderate depths in some veins. In at least one instance the downward continuation of the vein has been shown to be aplite. Other instances might be cited in which the aplite granite contact is marked by pegmatitic quartz, containing any of the following minerals: tourmaline, pyrite, sphalerite, or galena; occurrences in which the cores of aplite dikes are pegmatitic quartz with molybdenite and pyrite; and examples in which very small aplite dikes have been traced through sulphide-bearing aplite to veins.

All of these facts are mentioned in order to show that there may well be an aplite period of vein formation in Butte. But this in no way contradicts or modifies the conclusions of R. H. Sales,[4] as the aplite-age veins, if present, would merely be a portion of the country rock into which the later copper-bearing solutions were introduced.

Residual higher temperature ores have been found in the Meaderville portion of the copper area by Joseph Lyden. These earlier ores are characterized by tin-bearing tennantite and hübnerite with little else but quartz. Enargite is frequently found pseudomorphic after hübnerite.

Reply to W. E. Gaby.—Mr. Gaby expresses an opinion that the horizon of the Butte veins has been placed too near the top of the granite, and his sole argument is that he does not believe large inclusions or roof

[3] *U. S. Geological Survey, Bulletin* 620 (1916), 295.

[4] Ore Deposits at Butte, Mont. *Trans.* (1913), **46,** 3.

pendants of andesite exist in the district. Mr. Gaby, so far as we know, is alone in this belief. The evidence, both megascopic and microscopic, has without exception and in numerous places at Butte, shown the existence of andesite which is earlier than the rhyolite. At other places, farther from the mineralized district, this same andesite is seen in contact with granite and has been metamorphosed for a short distance from the granite contact. Recent very accurate surface mapping by M. H. Gidel and C. H. Steele, with which Mr. Gaby is not familiar, has been conclusive in showing the andesite-rhyolite relations.

The approximate horizon of the Butte veins, relative to the original surface of the granodiorite intrusive, is established by such additional evidence as:

1. Large diorite inclusions are found to the southwest of the productive area. These very large inclusions are found only near the original surface of the intrusion.

2. Sedimentary rocks, forming the cover of the batholith, are known at moderate distances to the east, south, and west, and show that the general horizon of the vein formation at Butte is near the original surface of the intrusive.

As will be noted, the authors of this paper have not given a definite figure for the depth of the outcrops below the cooling surface of the batholith, but the true figure, if it could be ascertained, might just as likely be less than the one given, as greater.

Reply to J. B. Hastings.—Mr. Hastings' question, "Where's the basalt," may be answered by the third paragraph, page 290: "Rare occurrences of late diabase of Pliocene age mark the last phase of igneous activity in this region."

That the aplite dikes, veins, and later faults tend to follow lines of least resistance will probably be conceded. That these lines of least resistance were determined by cooling phenomena, with accompanying tensional stresses, is the belief of the authors of this paper, because all the regional stresses subsequent to the granite intrusion have been tensional, and the regional stresses of compressive character have preceded the igneous activity. This gives the cycle: (1) compression; (2) adjustment by the transfer of liquid rock; (3) cooling and crystallization of the liquid rock, with shrinkage and tensional forces resulting in a general settling of the country in blocks. Small compressive forces are generated in the process of block settling, but these are in no way comparable with the regional stresses.

# A New Silicate of Lead and Zinc

BY P. A. VAN DER MEULEN,* PH. D., ITHACA, N. Y.

(St Louis Meeting, October, 1917)

SOME time ago, the writer received from W. O. Borcherdt, Superintendent of the works of the Bertha Mineral Co. at Austinville, Va., several specimens of a dense yellowish slag-like material, containing cavities showing clear needle-like crystals, and representing a product formed by the fumes of zinc oxide and basic lead sulphate from the oxide furnace, attacking the firebrick lining of the flues.

The matrix, which is of a greenish-yellow color, is made up chiefly of an amorphous glass, carrying numerous crystals, the latter being well-defined, clear, and transparent, but colored a very light yellow by small amounts of ferric oxide.

A number of the crystals were carefully separated from the matrix and tested with the blowpipe. They have a fusibility of about 2, yield a globule of lead and a coating of lead oxide before the blowpipe on charcoal, and contain some zinc. Their hardness is 5 to 6.

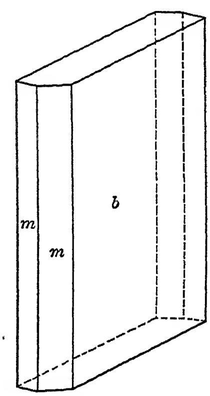


FIG. 1.

Under the microscope the crystals appear to be made up of a prism, with narrow faces, and a fairly large pinacoid, but unfortunately show no terminal faces (Fig. 1). Between crossed nicols they gave parallel extinction in the three planes containing the crystallographic axes, and are therefore orthorhombic. The index of refraction is high.

A number of the best crystals were measured on the reflection goniometer, the angles given below representing the average of measurements made on different individuals.

$$mm = 46° 52'$$
$$mb = 66° 34'$$
$$a:b:c = 0.4334:1:?$$

Inasmuch as the results indicate that the crystals do not correspond with any silicate of lead and zinc, either natural or artificial, heretofore

* Acting Assistant Professor in Mineralogy, Cornell University.

described, a complete chemical analysis of them was undertaken. An analysis of the matrix was also made.

The method employed in each case was briefly as follows: A weighed amount of the finely powdered substance was heated with dilute hydrochloric acid on the water bath, this treatment resulting in the complete decomposition of the crystals. The matrix left a slight undecomposed residue which was separated from the solution by filtration, and fused with a small amount of sodium carbonate, and then dissolved in dilute hydrochloric acid. The solution was evaporated to dryness and taken up in about 200 c.c. of warm water containing a few drops of hydrochloric acid, filtered while warm, and the filtrate again evaporated, diluted, and filtered to remove the last traces of silica. To the filtrate, dilute sulphuric acid was added and the solution evaporated until sulphur trioxide fumes were freely given off. It was then diluted with distilled water, and alcohol added to the amount of 50 per cent., after which the lead sulphate was filtered on a weighed Caldwell-Gooch crucible. From the filtrate, zinc was precipitated with hydrogen sulphide by the modified Waring method,[1] and determined as pyrophosphate in the usual manner. Iron and aluminum were removed by double precipitation with ammonia, and magnesium determined in the filtrate.

The results are given below.

| | Crystals | | | | Matrix, Per Cent. |
|---|---|---|---|---|---|
| | Per Cent | Ratio | | | |
| $SiO_2$ | 13.38 | 0.2218 | 1 | 2 | 7.11 |
| PbO | 60.37 | 0.2706 | Combined | | 54.25 |
| ZnO | 22.70 | 0.2789 | 2.477 | 5 | 32 51 |
| $Al_2O_3$ | 1.43 | | | | 3.55 |
| $Fe_2O_3$ | 1.53 | | | | 2.33 |
| CaO | none | | | | tr. |
| MgO | 0.87 | | | | 0.13 |
| Total | 100.28 | | | | 99.88 |

Leaving out of consideration the minor amounts of magnesia, alumina, and ferric oxide,[2] the crystals are seen to be a basic silicate of lead and zinc of the type $R''_5Si_2O_9$ in which $R''$ is replaced partly by Pb and partly by Zn, in nearly equivalent amounts. The density of the powdered crystals was determined by the pycnometer method and found to be 6.153 at 20° C.

[1] *Journal of the American Chemical Society* (1907), **29**, 264.

[2] If the iron oxide and alumina are calculated with the silica as acid-forming groups and the magnesia with the bases, the ratio of base to acid becomes 4.680 : 2 instead of 4.954 : 2, but it appears to the writer that these are admixed impurities and not part of the regular crystal compound.

The matrix is seen to be still more basic than the crystals, and probably contains some of the free oxide of lead or zinc or both, as a thin section of the matrix shows numerous needle-like crystals of the lead zinc silicate embedded in an amorphous groundmass.

In order to obtain crystals that might be measured and described more completely, a portion of the matrix was placed in a fire-clay crucible, melted in a gas crucible furnace and allowed to cool slowly. Crystals were readily obtained which could be identified under the microscope as being of the same kind as those described, but they were so small and poorly formed that it was not possible to measure them.

From a careful search of the literature, it does not appear that any basic silicate of lead and zinc, or of lead or zinc alone, of the type $R''_5Si_2O_9$ has been described heretofore, either as a mineral or as an artificial product. The system $PbO–SiO_2$ has been investigated thermally by Cooper, Krauss, and Klein.[3] They established the existence of the compounds $PbO.SiO_2$; $2PbO.SiO_2$; and $3PbO.2SiO_2$, and considered the existence of the compound $3PbO.SiO_2$ as probable, although no evidence of it was obtained in the melting-point curve. It seems not improbable, in view of the results of the analyses of the crystals described, that the compound $5PbO.2SiO_2$ should also exist and would probably be formed from a melt which is more basic than the crystals themselves.

## *Summary*

Crystals of a new silicate of lead and zinc obtained from a lead-zinc furnace slag are described. They are orthorhombic and made up of the prism and the brachypinacoid, and have an axial ratio 0.4334 : 1 : ?.

Chemical analysis shows the composition of the crystals to be represented by the formula $R''_5Si_2O_9$ in which $R''$ represents lead and zinc in nearly equivalent amounts. They contain small amounts of magnesia, iron oxide, and alumina.

These crystals can be produced by the slow cooling of a matrix which is more basic than the crystals themselves.

[3] *American Chemical Journal* (1910), **47**, 273–85.

# The Effects of Cross Faults on the Richness of Ore

BY EDGAR K. SOPER, PH. D.,* CORVALLIS, OREGON

(St Louis Meeting, October, 1917)

## Introduction

It has been observed that where veins or other types of orebodies are intersected by cross faults, the continuation of the ore deposit below the fault is often of lower grade than that portion above the dislocation.[1] In some mines, the reverse of these conditions has been noted, and the richest ore is found to occur in that portion of the vein *beneath* the fault intersection. Such instances seem to be less frequent than those where the richest ore is *above* the offset. The important and widespread influence of fault and vein intersections upon the localization of ore shoots is well known,[2] but the factors that govern the position of the ore shoot relative to the fault intersection seem to be less clearly understood. The richest ore may be above or below the fault, or it may occur at the point of intersection, or there may be no change.

Recent observations made by the writer in mines in Idaho and Montana suggested a possible explanation for the changes in the richness of ore below certain types of fault intersections. These observations have been supplemented by a study of the literature containing detailed descriptions of ore deposits at a large number of mines in North America. The results of these studies are presented herewith in the hope that they may lead to further discussion of this important phase of ore deposition.

## Relative Age of Faults

Faults which intersect veins and other ore deposits may be either pre-mineral or post-mineral in age—that is, they may be older or younger than the deposits which they intersect. Many veins are formed along

* Dean, Oregon School of Mines.

[1] Effects of Faults. *Mining and Scientific Press* (Dec. 23, 1916), **113**, 902; (Feb. 3, 1917), **114**, 152.

[2] W. Lindgren: *Mineral Deposits* (1913), 176, N. Y., McGraw-Hill Book Co.

S. F. Emmons: Structural Relations of Ore Deposits. *Trans.*, Ore Deposits Volume (1913), 26–64.

fault fissures, while many others have been reopened by post-mineral faulting. In some veins, as at Butte, Mont.,[3] there is a record of several successive periods of movement along the veins, followed in each case by a healing up of the fissure by mineral-bearing solutions, and resulting in several periods of ore deposition within the same vein.

Veins that are younger than faults often end abruptly against the faults. No continuation of such a vein is to be expected beyond the fault. Cross faults, or faults that cut across the veins in a mining district, are usually younger than the veins, although many times the continuation of a vein beyond the fault is never found. Cross fissures are often contemporaneous in age with the vein fissure. J. E. Spurr[4] has recently discussed the close relation between post-mineral faulting and igneous intrusions in mining districts. Spurr points out that much of the post-mineral and post-intrusion faulting in mining regions may be due to adjustment strains consequent upon the consolidation of magmas, and that the overlying rocks are repeatedly adjusted along fault planes.

The frequent formation of veins along fault fissures is well known, and the attendant phenomena have been fully discussed by others.

It is not intended in this paper to consider the well-known effects of post-mineral faults which follow veins, nor veins which follow pre-mineral faults. The discussion is limited to the effects on the richness of ore produced by faults which intersect and dislocate orebodies. Such cross faults must all be post-mineral in age—that is, younger than the veins and orebodies which they offset, for, obviously, the latter must have existed when the faulting occurred which caused their dislocation.

The relative age of a fault with respect to veins which it intersects, or cuts off, is most important in determining the effect of the fault upon the richness of ore in the veins, and the position of existing ore shoots. Pre-mineral faults may limit a vein so that the vein lies entirely above or below the fault. There are many examples of this throughout the mining regions of the country.

In the case of post-mineral cross faults, which offset veins and other ore deposits, the effects produced on the richness of the ore on opposite sides of the fault plane become more evident. These effects are discussed in the following pages, and examples are cited where such effects may be observed.

The criteria for detecting the relative age of faults and veins which

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[3] W. H. Weed: Geology and Ore Deposits of the Butte District, Montana. *U. S. Geological Survey, Professional Paper* 74 (1912).

R. H. Sales: Ore Shoots at Butte, Montana. *Economic Geology* (1908), **3,** 326–331.

[4] J. E. Spurr: Relation of Ore Deposition to Faulting. *Economic Geology* (1916), **11**, 619.

they intersect are discussed by Ransome[5] in his report on the ore deposits of the Coeur d'Alene district, Idaho.

## Types of Faults

Strike faults, dip faults, and oblique faults may intersect veins, and either cut them off abruptly, or dislocate them, producing an offset in

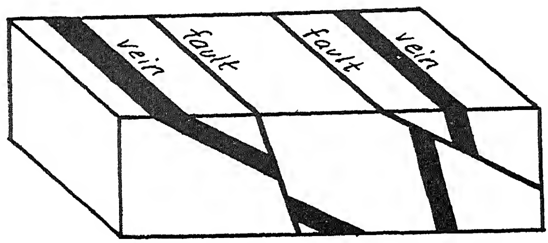


FIG. 1.—STRIKE FAULTS DIPPING IN SAME DIRECTION AS VEIN.

the vein. A strike fault, dipping in the same direction as the vein, may cut and offset the vein, if at the surface the fault lies in the foot-wall side of the vein, and dips toward the vein at an angle less than the dip of the vein (see Fig. 1).

If the fault lies in the hanging-wall side of the vein at the surface, it will cut the vein if its dip is steeper than that of the vein (Fig. 1). A

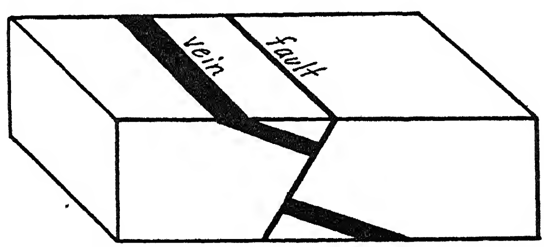


FIG. 2.—STRIKE FAULT DIPPING IN OPPOSITE DIRECTION TO VEIN.

strike fault dipping in a direction opposite to that of the vein must necessarily intersect the vein regardless of the dip, if it lies in the hanging-wall side of the vein (see Fig. 2).

In so far as the effect of the fault upon the richness of ore is concerned, it makes no difference whether the fault is of the normal or reverse type. The important feature is the fact that wherever a strike fault offsets a

[5] F. L. Ransome: Geology and Ore Deposits of the Coeur d'Alene District, Idaho. *U. S. Geological Survey, Professional Paper* 62 (1908), 120–121. Also F. L. Ransome: The Relations between Certain Ore-Bearing Veins and Gouge-Filled Fissures. *Economic Geology* (1908), **3**, 331–337.

vein there is always a segment of the vein below the fault. In referring to locations below, or beneath a fault, that portion of the vein is meant which lies down the dip of the vein and beyond the fault intersection. Even when the fault is vertical, the continuation of the vein along its dip beyond the fault intersection may be said to be below the fault.

When a dip fault intersects a vein, that segment of the vein lying in the foot-wall side of the fault may be spoken of as lying below the fault

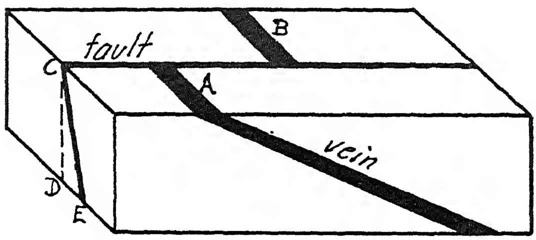


FIG. 3.—DIP FAULT, WITH STRIKE PARALLEL WITH DIRECTION OF DIP OF VEIN.

(see Fig. 3). Where the fault plane is vertical, there is no foot wall or hanging wall with reference to the fault, and hence, this rule fails. When a fault cuts a vein at an angle oblique to both the strike and dip of the vein, it is called an oblique fault (Fig. 4 and 5). The great majority of faults are of this type. In veins which have been offset by oblique faults, that segment of the vein which lies on the foot-wall side of the fault may be said to be below the fault (see Fig. 4 and 5).

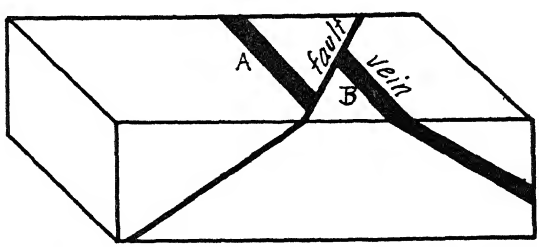


FIG. 4.—OBLIQUE FAULT, DIPPING IN OPPOSITE DIRECTION TO VEIN.

From a study of the accompanying diagrams (Fig. 1 to 5), it will be seen that the segments of veins below the intersections of dip and oblique faults are, in reality, not below the faults in the same sense as in the case of strike faults. This may be seen more clearly by tracing the path followed by solutions which may be assumed to be tending to flow from the outcrop downward along the dip of the vein.

In Fig. 1 and 2, surface waters seeping into the vein along the outcrop at any point, and following down the dip, must inevitably en-

counter the strike fault at depth regardless of the dip of the fault or vein. Thus all of the segment of the vein beyond the fault intersection is below the fault. Such is not the case with dip faults and oblique faults, as may be seen by reference to Fig. 3, 4, and 5.

In the case of dip faults (Fig. 3), solutions may enter the outcrop of the vein on either the hanging or foot-wall side of the fault. Waters seeping down the dip of the vein on the foot-wall side of the fault can never encounter the fault. On the hanging-wall side of the fault, such solutions may or may not encounter the fault, depending upon the dip of the fault and the depth to which the solutions descend along the vein. Thus, in the case of dip faults, only that portion of the vein which lies in the long wedge-shaped block of ground *CDE* on the foot-wall side of the fault may be said to lie below the fault.

The case of oblique faults is somewhat similar to that of dip faults, with the difference that there is usually a greater portion of the vein below the fault. In Fig. 4, assume that surface water enters the vein along the outcrop on the hanging-wall side of the fault, at *A*, and seeps downward along the direction of dip. The solutions will soon encounter

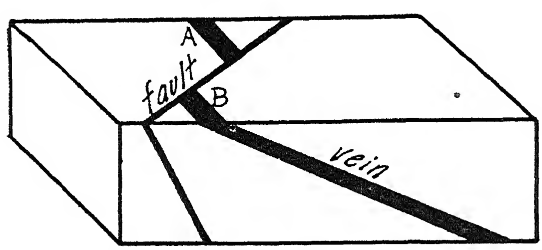


FIG. 5.—OBLIQUE FAULT DIPPING IN SAME DIRECTION AS VEIN.

the fault. But if surface waters enter the outcrop of the vein on the foot-wall side of the fault, at any point *B*, and travel downward along the dip of the vein, they will never encounter the fault. Likewise, in Fig. 5, solutions entering the vein at *A* and traveling down along the dip of the vein must encounter the fault, provided they descend to sufficient depth. Waters finding entrance to the vein at outcrop *B* (Fig. 5) and traveling downward on the dip will not encounter the fault.

It is well known, however, that waters descending along veins and fissures do not necessarily follow the direction of dip, but often follow more permeable paths oblique to the dip of the fissure. Under such conditions, for example, waters entering the vein at *B*, Fig. 5, might pass downward along the vein along the intersection between the vein and the fault. In fact, this intersection would form a natural trough along which descending solutions might easily find a way.

## Ores Commonly Affected

The change in the nature of ores below fault intersections is most pronounced, and has been most frequently observed, in copper deposits. Deposits of silver and zinc ores also show marked differences in character and grade on opposite sides of fault intersections. Gold-bearing ores have also been noted which carry diminished values below faults. Lead ores, on the other hand, do not, as a rule, exhibit such changes where the deposits are cut by faults. The effects of faults on the richness of ores of the rarer metals have not been investigated by the writer, and search through the literature has failed to disclose sufficient data to justify definite conclusions. However, the variation in the effects produced by faults on the ores of the five metals named above is very suggestive. It has been noticed that copper ores, which are very soluble in the oxidized zone, and therefore most subject to leaching and secondary sulphide enrichment, exhibit the most pronounced changes in richness below faults. On the other hand, lead ores, particularly galena, which are most stable under surface conditions, and which seldom exhibit important secondary sulphide enrichment, show the least variation in richness on opposite sides of cross faults.

In general, it may be observed that ore deposits which show important secondary enrichment in the upper levels are liable to become lower in grade below post-mineral cross faults. Deposits consisting essentially of primary minerals may or may not show a change in tenor below faults. If primary ores do show such a change in values, the rich ore may be either above or below the fault. Where veins are terminated abruptly by faults, and the continuation of the veins are not found, the faults may be either pre-mineral or post-mineral. If the faults are pre-mineral in age, the veins will probably have no continuation beyond the fault planes.

## Effects Produced in Veins

Most of the data collected concerning the effects of faults upon the richness of ore has been obtained from a study of vein deposits. Veins constitute the commonest type of ore deposits, and they furnish the most conclusive evidence regarding the effects of faults.

Where veins are intersected and offset by faults, the ore beneath the plane of dislocation is often of lower value than that above the fault. The rich ore above the fault usually lies in a well-defined ore shoot, or in a series of shoots, the remainder of the vein being of lower grade.

These ore shoots often occur just above, and in close proximity to the fault zone, but they may be encountered at any horizon between the outcrop and the fault intersection. The occurrence of rich ore shoots just above the fault plane is often due to the fact that the intersection of the two fissures, the vein and the fault, is especially favorable to ore

enrichment. The cold, enriching, descending solutions from the upper part of the vein may be dammed back temporarily by the gouge of the fault zone, and in copper, silver and gold deposits, exceptional enrichment may result in such places. Galena is seldom secondary, and hence in lead deposits secondary sulphide enrichment is of far less importance than in copper, zinc, silver and gold deposits, although important enrichment through oxidation may take place in the upper portions of galena veins.

In the Coeur d'Alene district in Idaho, most of the lead-silver deposits are essentially galena veins. In some of these veins, for example in the Hypotheek, oxidation is important to considerable depths but the big ore shoots of the district consist of primary galena.

Ore shoots just above the fault plane are also frequently due to reactions set up by the mingling of solutions from the vein and those traveling along the fault. Reactions between the solutions descending along the vein, and the fault gouge may also cause the precipitation of ore minerals. Such ore shoots are often the result of secondary sulphide enrichment by cold descending solutions. The solutions passing downward along the vein may encounter the fault and thus be deflected from their downward course along the vein to a new course along the fault, which may be more permeable than the vein. This would limit secondary enrichment to that portion of the vein above the fault intersection, and explain the absence of high-grade ore below that horizon in many deposits of this type. Whether these solutions would encounter the fault and be deflected would depend chiefly upon the relative positions of the vein and fault as previously discussed and as illustrated in Fig. 1 to 5.

In support of this statement may be cited numerous instances where the ore shoots above the fault consist chiefly of secondary sulphide minerals or minerals characteristic of the zone of oxidation, while the ore in the offset segment of the vein below the fault intersection is made up of the lower-grade primary sulphides, and well-defined ore shoots are wanting.

At Butte, Mont., the writer has observed several instances where cross faults have caused enrichment to be confined to the portions of the veins above fault intersections. Weed[6] showed that the cross faults in the Butte district often deflected the downward moving solutions in the east-west veins and caused them to descend along the cross faults or to spread out over the clay walls of the faults and form ore shoots. He also cites instances where both cross faulting and strike faulting have opened primary quartz-pyrite veins, thus permitting the descent of water down the vein, which resulted in secondary enrichment above the cross fault.[7]

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[6] W. H. Weed: Geology and Ore Deposits of the Butte District, Montana. *U. S. Geological Survey, Professional Paper* 74 (1912), 103.

[7] *Op. cit.*, 110.

At Butte, the effect of cross faults (they may be dip faults, strike faults, and oblique faults) on ore deposition is also well shown, even where there is no clay gouge to deflect enriching solutions, in the frequent "spraying out" or widening of the veins just above a fault intersection.[8] In most of the Butte mines there is a marked tendency for ore shoots to be localized at or near cross fractures or faults. The ore shoots are usually above the fault intersections and frequently these cross faults mark abrupt changes to lower-grade ore below. Occasionally they may cut off the ore entirely. The occurrence of rich shoots of secondary ore just above cross faults and the change to low-grade ore below the faults is mentioned by Weed[9] in numerous places in his report on the Butte district.

Ransome[10] noted the effect of cross faults on the richness of ore in some of the gold-bearing veins of Farncomb Hill in the Breckenridge district of Colorado. He says that the ore shoots usually occur above cross faults, which, in some places, divide oxidized ore above the fault from a calcite vein below.

At Rossland, B. C., the localization of ore shoots is often controlled by cross faults or dikes. An example, cited by C. W. Drysdale,[11] from the fourth level of the Center Star mine, shows where good ore ends abruptly at the intersection of the vein with a dike, although the primary sulphides in the vein continue uninterruptedly beyond the dike. The dikes often show slip planes along their walls. In many other instances, according to Drysdale, ore shoots terminate against faults, on the upper side of which they sometimes increase enormously in width. In still other instances at Rossland,[12] ore shoots occur at the intersection of cross fractures beyond which the ore is of low grade. These cross fractures may or may not show slipping, but in so far as their effect on the richness of the ore in the vein is concerned, they act essentially like faults.

In the Coeur d'Alene[13] district of Idaho, although cross faults often cut off the veins, the influence of the faults upon the richness of ore is not so apparent. This lack of influence, however, might be expected, since the principal ore mineral of the Coeur d'Alene deposits is galena, which is undoubtedly primary. Very little oxidation has taken place except near the surface, and secondary sulphide enrichment of the lead-silver

[8] *Op. cit.*, 112, 121.

[9] *Op. cit.*, 114, 121, 122, 123, 124, 125, 126, 130, 131, 132, 133.

[10] F. L. Ransome: Geology and Ore Deposits of the Breckenridge District, Colorado. *U. S. Geological Survey, Professional Paper* 75 (1911), 157.

[11] C. W. Drysdale: Geology and Ore Deposits of Rossland, B. C. *Canada Geological Survey, Memoir* 77 (1915), 63.

[12] *Op. cit.*, 64.

[13] For detailed description of the Coeur d'Alene ore deposits see F. L. Ransome: The Geology and Ore Deposits of the Coeur d'Alene District, Idaho. *U. S. Geological Survey, Professional Paper* 62 (1908).

deposits is insignificant. Consequently, cross faults have produced little or no apparent change in the tenor of the veins. However, most of the veins in the Coeur d'Alene district occur along fault fissures, or fault zones. The ore has been deposited both by replacement of the rock along the fault zone and by deposition in open places. In this respect, faulting has been of the utmost importance in the Coeur d'Alene.

The lead-silver deposits of central and southern Idaho have not yet been developed to depths comparable to the depths obtained in the Coeur d'Alene district in the northern part of the State. The deposits in the central and southern region often show oxidation to considerable depths. In fact, many of the deposits have not been developed below the oxidized zone. In a number of mines in this region, the orebodies have been cut off by post-mineral faults. Often the veins were never found beyond the faults, and hence no data are available as to what effect the faults had on the richness of the ore. In a few cases where data are available, the cross fissures seem to influence the development of ore shoots as in the Texas district in Lemhi County.[14] It is probable that some of these cross faults belong to the period of mineralization and occurred before the veins were formed. A few others seem to be post-mineral and they have shut off oxidation and enrichment from the vein segments below the intersection.

An example showing conclusive evidence of the effect of post-mineral cross faults on the richness of ore may be seen in the Pittsburg-Idaho mine, at Gilmore in the Texas district. In this mine nearly all the ore developed to date has been almost completely oxidized. However, a cross fault with a heavy gouge seam cuts the vein. Below this fault were the only stopes of primary ore in the mine prior to 1913.[15] The ore there contained considerable argentiferous galena with some pyrite. Above the fault the ore was completely oxidized.

Another excellent example of the effect of a cross fault in causing the ore below the fault to be of lower grade than that above may be seen in the Clipper Bullion mine in the Mineral Hill district, Lemhi County, Idaho.[16] The ore deposit, which is mined for gold, is a quartz vein. In the upper part the vein consists of decomposed honeycomb quartz carrying free gold. The vein is faulted at short intervals but the offsets are only a few feet. Above one of these cross faults (a strike fault) the ore was completely oxidized, and below it unaltered primary ore, consisting of auriferous pyrite irregularly scattered in a quartz-barite gangue, was encountered for the first time.

At Goldfield, the orebodies have not been modified to any important

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[14] J. B. Umpleby: Geology and Ore Deposits of Lemhi County, Idaho. *U. S. Geological Survey, Bulletin* 528 (1913), 94.

[15] *Op. cit.*, 96.

[16] *Op. cit.*, 144.

extent by post-mineral faulting.[17] It is difficult and often impossible to secure conclusive evidence as to the relation of the faults to the ledges because of the extreme irregularity of the orebodies themselves. In a few cases cross faults of small throw have been recognized, which cut off the ore. One of the most important of these is seen in the Florence mine, where a gouge-filled fault fissure cuts off the ore on the 100-ft., 200-ft. and 350-ft. levels.[18] However, the continuation of the ore beyond this fault has not been found and hence no evidence is available that would indicate the effect of the fault on the richness of the ore.

The ore deposits of Tonopah, Nev., are remarkable for the complex faulting which they have undergone. The productive part of the district lies in a rectangular fault block, bounded on all four sides by faults which cut off the veins. Within this productive area are a great number of minor faults and cross fissures which intersect the veins. Many of these cross fissures are pre-mineral in age, and although they have produced no offsets in the ore deposits, they have been most important in controlling to a large extent the localization of certain ore shoots.[19] Most of the minor faults within this block are post-mineral in age, and have produced numerous, although small, offsets in the vein. Step faulting is common. Cross veins of the same age as the main east-west veins are also numerous, and these often cut off the ore in the main veins, although there is no offset.[20] Although some of the veins at Tonopah have been offset again and again by cross faults, there does not seem to be any important change in the richness of the ore below these faults. The various segments of these faulted veins are apparently fairly uniform in value. The explanation probably lies in the fact that the ore at Tonopah is primary. The mineralization was complete before the veins were faulted, and since the faulting occurred there has been but little secondary alteration of the metallic minerals in the veins, and the distribution of values has remained unchanged.

At Morenci, Ariz., the great faults are younger than the veins, for they usually dislocate them. Lindgren,[21] in his report on the Clifton-Morenci district, says that in the Manganese Blue mine the ore deposits were cut off on the east by the Copper Mountain fault which is later than the primary sulphide ore, but the oxidation of the sulphides has taken place since the fault was formed. Very little ore was found east

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[17] F. L. Ransome: Geology and Ore Deposits of Goldfield, Nevada. *U. S. Geological Survey, Professional Paper* 66 (1909), 161–162.

[18] *Op. cit.*, 227.

[19] J. E. Spurr: Geology of the Tonopah Mining District of Nevada. *U. S. Geological Survey, Professional Paper* 42 (1905), 119.

[20] *Op. cit.*, 130.

[21] W. Lindgren: The Copper Deposits of the Clifton-Morenci District, Arizona. *U. S. Geological Survey, Professional Paper* 43 (1905), 208, 248–250 and 275.

of the fault, and this was composed largely of primary sulphides too low-grade to work, while the vein above the fault contained much rich oxidized ore. In the Humboldt mine, in the same district, the Copper Mountain fault separates good milling ore on the west from barren porphyry on the east, or low-grade primary ore. Other mines of the district show faults which have exerted an important influence upon the location and richness of the ore deposits.

Many additional examples could be cited of veins that have been displaced by post-mineral faults resulting in a difference in the richness of the ores in the various segments of the vein. The cases enumerated, however, will illustrate the manner in which these changes are brought about.

## Effects on Deposits Other than Veins

The effects produced by faults on the richness of ore in deposits other than veins, such as bedded, contact, replacement and disseminated deposits, are not to be compared in importance with the effects produced in fissure veins and lodes. Such irregular replacement deposits do not, as a rule, have direct channels connecting with the surface along which descending solutions might find their way, comparable to the pathways for such solutions furnished by fissure veins. If post-mineral faults cut irregular deposits of the various types just named, the fault plane itself would usually constitute the most direct route to the ore for descending solutions. Thus, instead of deflecting the enriching solutions away from the deposit, as is often the case where a fault dislocates a fissure vein, the fault may be the chief source of supply for these solutions. The intense alteration of both ore and country rock which often occurs all around the fault zone in replacement and contact deposits supports this statement. The alteration often obliterates all evidence as to the relative age of the fault and vein. The changes in the richness of ore in deposits of this class are less conspicuous and more difficult to detect than in fissure veins. The great irregularity of form so often exhibited by the deposits renders the evidence unconvincing.

Post-mineral faults are often of less importance in their relations to irregular replacement deposits than are pre-mineral faults. The latter are often the principal channels along which solutions traveled which produced the original mineralization.

Irving[22] has shown how faults may be preserved in replacement deposits. If the ore is not itself affected by the displacement, it is younger than the fault. Replacement ore which is displaced and offset is older than the fault, but it is often difficult to determine whether replacement ore has been offset.

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[22] J. D. Irving: Replacement Orebodies. *Economic Geology* (1911), **6**, 642–3.

At Morenci, the faults which played such an important part in controlling the distribution of the values in the veins and lodes were also influential in controlling the relations between richness of ore and depth, in the contact and disseminated deposits.[23] Many of the faults in this district took place after the chalcocite was formed. In such cases, therefore, there is no such pronounced change in the richness of the ore below the fault intersection, since the ore was enriched by chalcocitization previous to the faulting. On the other hand, oxidation of the Morenci-Metcalf ores was largely later than the faults. This may explain the occurrence of rich oxidized ore above fault planes, in some cases actually abutting against the faults, and the change to lean primary milling ore in the segments of the orebody below the fault.

In the Globe copper district, Arizona, there are hundreds of faults within the mineralized area. While the major faults of the district are pre-mineral, there are numerous post-mineral faults, many of which have cut off or displaced the orebodies, which are chiefly irregular deposits in limestone. In the Old Dominion mine, for example, the large orebody above the second level was cut off abruptly by the Alice fault,[24] which is one of a number of a complex zone of intersecting post-mineral faults. The continuation of the ore deposits thus cut off may or may not be richer than the ore previously stoped. The relations at Globe are complicated by the fact that the ore was formed and enriched by oxidation and chalcocitization before the dacite flow occurred which buried the oxidized outcrops, and the faulting is even later than the dacite. Hence the segment of the orebody on the downthrow side of the fault may contain oxidized and enriched ore at depths considerably below the sulphide horizon in the portion of the deposit above the fault.

At Bisbee,[25] Ariz., faulting was also one of the chief factors in controlling the distribution of the orebodies, but the influence has been due to the fact that the major faults in the vicinity of the large orebodies are pre-mineral and served as channels along which the solutions traveled which caused much of the primary mineralization. The same faults acted as important agents in permitting oxidation of the primary ore at a later date. Bisbee does not furnish any clear examples of offset ore shoots which are of lower grade below faults than above.

## Summary

1. Ore deposits which have been intersected by cross faults often consist of lower-grade ore in the segments of the deposits below the dislocation than above.

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[23] W. Lindgren: *Op. cit.*

[24] F. L. Ransome: Geology of the Globe Copper District, Arizona. *U. S. Geological Survey, Professional Paper* 12 (1903), 144–145.

[25] F. L. Ransome: The Geology and Ore Deposits of the Bisbee Quadrangle, Arizona. *U. S. Geological Survey, Professional Paper* 21 (1904).

2. Such changes in the tenor of the ore are most common in veins and lode deposits, where the fault intersection often marks the change from rich ore above to lean unprofitable material below.

3. Where such a change in the value of the ore exists, the relatively rich ore above the fault usually consists largely of oxidized material, or secondary sulphides, or both, whereas the ore below the dislocation, in typical cases, consists chiefly of the primary sulphides.

4. Under certain conditions cross faults do *not* affect the richness of ore below the intersection. The principal conditions under which no changes occur are:

(*a*) Where the faults are older than the ore, and hence cut off the ore completely, and no continuation of the orebody existed beyond the fault.

(*b*) Where faulting is post-mineral in age, but so recent that sufficient time has not elapsed for secondary alteration to enrich the ore above the fault.

(*c*) Where oxidation and secondary enrichment was well advanced or complete before faulting occurred so that the segments of the vein above the fault will not be subjected to any further alteration, and the segment below the fault will be insulated from alteration.

(*d*) Where the fault cuts the ore at such great depth that the intersection is well below the zone of secondary enrichment, and in primary sulphides of approximately uniform value.

(*e*) Where the ore consists of minerals highly resistant to secondary alteration such as galena and native gold.

5. While cross faults often exert important effects on ore deposits other than veins, such as replacement, contact, bedded, and disseminated deposits, such faults do not as a rule produce the marked changes in the tenor of the ore which are often brought about in veins. Pre-mineral faulting is probably just as important as post-mineral faulting in the enrichment of irregular replacement deposits.

6. The effects of cross faults on the richness of ore in veins are likely to be more pronounced in the case of strike faults and least pronounced in the case of dip faults. Oblique faults which intersect veins at acute angles to the strike of the veins are more likely to produce important changes in the ore than those which intersect veins at large angles.

7. The explanation for the frequent decrease in the value of ore below cross faults probably lies in the fact that the ore in the portions of veins above fault intersections has often undergone enrichment by cold descending solutions. These solutions, upon encountering the fault, would usually be diverted from the vein to the fault zone, preventing the enrichment of that portion of the vein below the dislocation.

# The Replacement of Sulphides by Quartz

BY H. N. WOLCOTT,* B. S., A. M., BISBEE, ARIZ.

(St. Louis Meeting, October, 1917)

AMONG the many cases of replacement of one mineral by another, that of quartz or silicates by pyrite, or even other sulphides, is not uncommon, but the reverse of this process does not appear to have been recorded, even if it has been observed. It seemed to the writer, therefore, that the case described below was worthy of record.

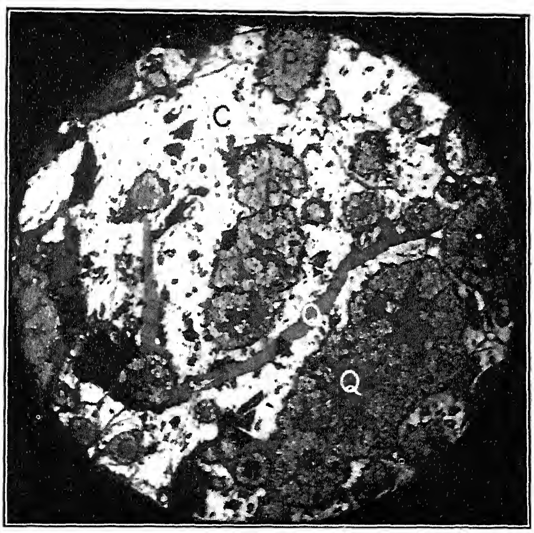


FIG. 1.—GROUNDMASS OF CHALCOPYRITE (C) CONTAINING ROUNDED GRAINS OF PYRITE (P) SHOWING DIFFERENT DEGREES OF REPLACEMENT BY QUARTZ (Q), AND ALSO QUARTZ VEINLETS CUTTING THROUGH THE PYRITE. (PHOTOGRAPHED BY REFLECTED LIGHT.)

The specimens described in this paper were obtained by R. B. Morton of Idaho Springs, Colo., from the Old Town mine, in Russell Gulch, and are in the collection of Cornell University, where the writer studied them.

In the rough, the specimens appeared to consist largely of chalcopyrite, with some grains of tetrahedrite. The polished surface of some of

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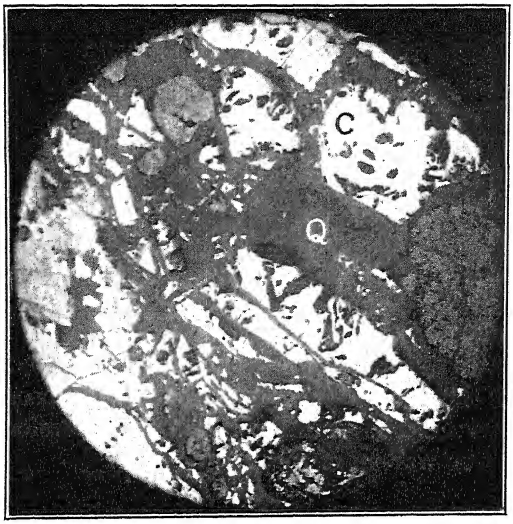


FIG. 2.—BRECCIATED ORE SHOWING CHALCOPYRITE (C), PYRITE (P) AND QUARTZ (Q). (PHOTOGRAPHED BY REFLECTED LIGHT.)

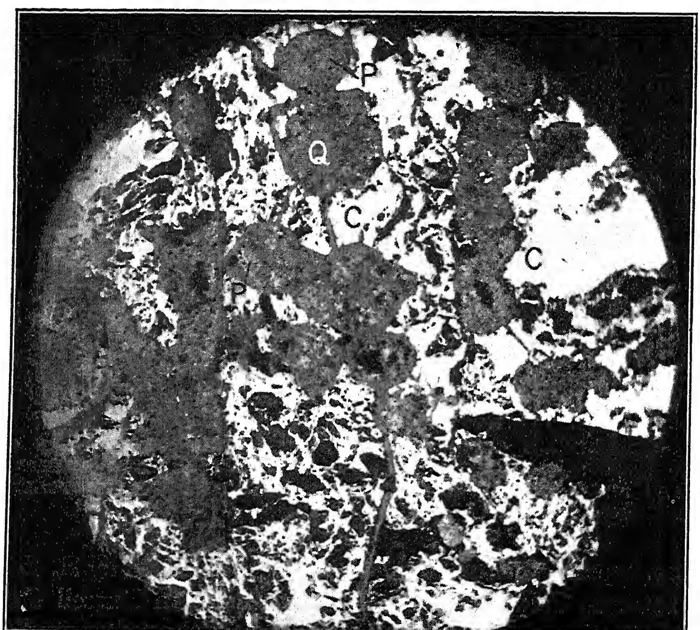


FIG. 3.—CHALCOPYRITE (C), CONTAINING PYRITE (P), AND VEINLETS OF QUARTZ (Q). THE LAST NAMED NOT ONLY CUTS THE CHALCOPYRITE, BUT REPLACES IT AROUND SOME OF THE PARTLY REPLACED PYRITE GRAINS, AS SHOWN BY CRYSTAL OUTLINES OF QUARTZ (METACRYSTS.) BLACK SPOTS ARE DEPRESSIONS IN SURFACE. (PHOTOGRAPHED BY REFLECTED LIGHT.)

the lumps, however, showed numerous rounded grains of pyrite, and minute fractures.

These smooth surfaces studied by reflected light under the microscope exhibited most interesting relationships. There, it was seen that while chalcopyrite and tetrahedrite were probably contemporaneous, numerous grains and rosettes of pyrite of varying size were scattered through the chalcopyrite. A few were seen in the tetrahedrite. As the pyrite is completely surrounded by the chalcopyrite (and also the tetrahedrite) it may be of earlier age.

The pyrite rosettes and grains consist in some instances entirely of that mineral. In other cases, and by far the most numerous, the rosettes show an irregular central mass of quartz (Fig. 1), of varying size, and often connecting with the exterior by one or more veinlets of quartz (Fig. 2).

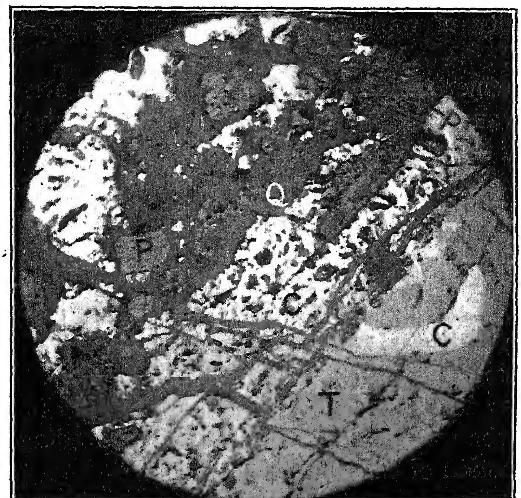

FIG. 4.—CHALCOPYRITE (C), AND TETRAHEDRITE (T), CUT BY QUARTZ VEINS PYRITE (P), IN PART REPLACED.

Some show a little quartz on the outside of the pyrite grain, and in one case at least (Fig. 3), the quartz surrounding the pyrite showed the crystal outlines of quartz.

The quartz, in addition to its occurrence in the pyrite grains, fills numerous narrow fractures in the chalcopyrite and tetrahedrite, indeed in some parts of the specimen, the chalcopyrite and tetrahedrite have been crushed to a mass of angular fragments (Fig. 2) with quartz filling the crevices so formed. One other point to be noted is that the quartz veinlets passing through the chalcopyrite and tetrahedrite are usually clean cut, and form connecting lines between the quartz in the pyrite rosettes. In places, however, the quartz veins enlarge and project into the chalcopyrite and tetrahedrite in such an irregular way as to suggest replacement.

Some of the areas in the tetrahedrite might be regarded as contemporaneous with the sulphide were it not for their connection with the feeding channels.

It seems to the writer that the structural conditions and mineral relationships described above lead necessarily to the following conclusions:

Following the deposition of the sulphides, fracturing of the ore occurred, resulting in the introduction of silica that was deposited in the chalcopyrite and tetrahedrite largely as fracture filling, but in the chalcopyrite at least in part by replacement. In addition it was deposited in the pyrite almost wholly by replacement.

The reasons for believing the above to be the case are that:

1. In every specimen examined the pyrite seemed to be earlier than the chalcopyrite, or in some cases contemporaneous with it.

2. The quartz forms veinlets which cut all the sulphides, thus precluding the possibility of replacement of quartz by pyrite or the other sulphides.

3. Clearly defined channels or feeders of quartz often lead from one pyrite rosette to another, showing the lines along which the silica-bearing solutions entered the ore.

4. The pyrite rosettes show all stages in the replacement process, but curiously enough the change seems to have worked from the center outward.

5. It does not seem likely that the pyrite could have been introduced with the quartz, otherwise one would expect to find deposition of pyrite along the feeding channels, whereas this is not the case.

6. The development of the crystal outlines of quartz in the chalcopyrite in places, indicates clearly a replacement of the chalcopyrite by the quartz, and the association of these with the feeding channels shows the later introduction of the material.

# Geology and Mineral Deposits of the Ozark Region

BY H. A. BUEHLER,* ROLLA, MO.

(St Louis Meeting, October, 1917)

## Location

THE Ozark region occupies a large part of the southern half of Missouri, the northern portion of Arkansas and comparatively small areas in northeast Oklahoma, southwest Kansas, and southern Illinois. It is a roughly elliptical plateau which has been deeply dissected by stream erosion, and which slopes imperceptibly into the prairie regions to the north, east, and west, but which terminates rather abruptly along the Arkansas River on the south and Tertiary lowland to the southeast. The Boston Mountains extending for a distance of some 200 miles (321 km.) along the Arkansas River rise to an elevation of 2000 ft. (609 m.), forming the most prominent topographic feature of the region. The so-called St. François Mountains in St. François, Madison, and adjoining counties in southeast Missouri, form a prominent but more restricted topographic feature, the highest point rising to an elevation of 1800 ft. (548 m.). The remainder of the region is largely a plateau area varying from 800 to 1500 ft. (243 to 457 m.) in elevation.

## Stratigraphy

The region is underlain chiefly by sedimentary formations, which outcrop in roughly concentric bands around a comparatively restricted area of igneous rocks in the St. François Mountain district. Although located geographically near the eastern edge of the Ozarks, the St. François Mountains have apparently suffered the greatest orogenic movement and the lowest formations of the succession are exposed in their immediate vicinity.

The various sedimentary formations of the region do not show any considerable metamorphism and have suffered little change since consolidation. With the exception of a few basic intrusions in Ste. Genevieve County and a pegmatite dike in Camden County, Missouri, there is no evidence of igneous action, since the first sedimentary beds were deposited.

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* State Geologist.

The accompanying columnar section (Fig. 1) indicates the chief geologic divisions and shows the stratigraphic horizon of the most important ore deposits of the region.

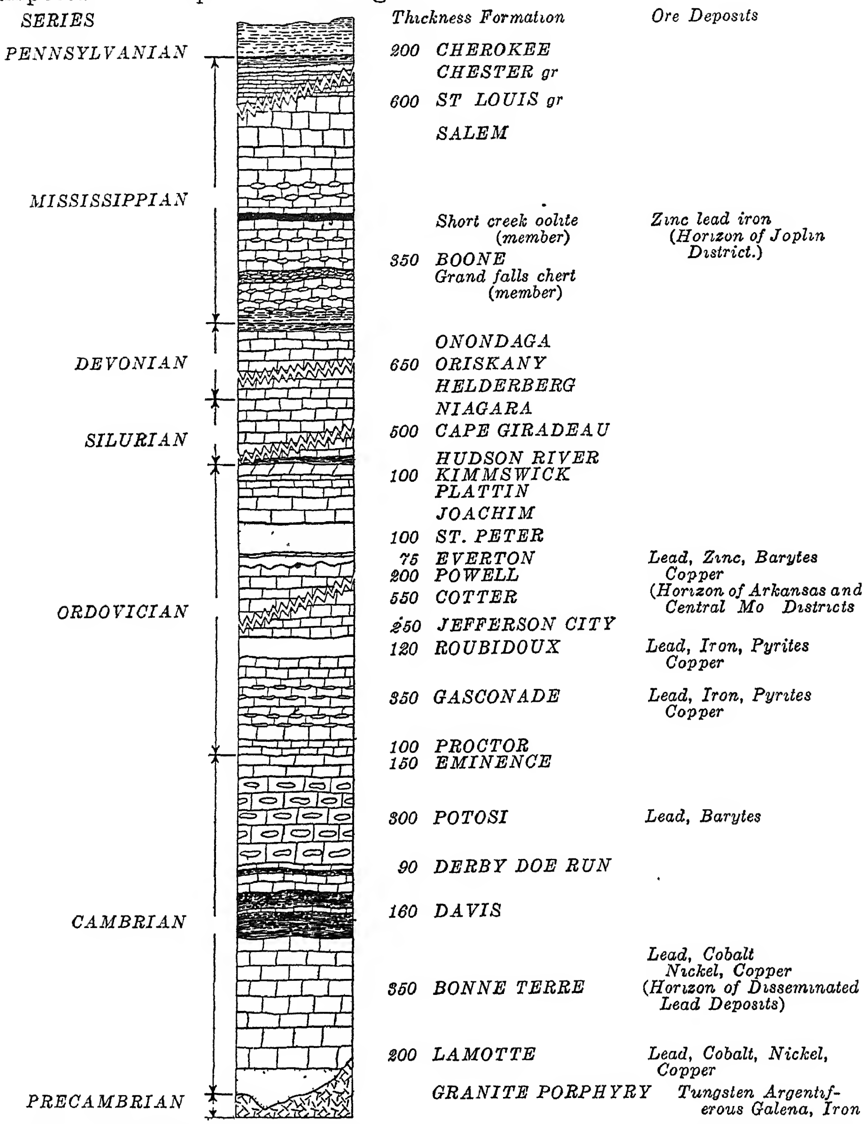
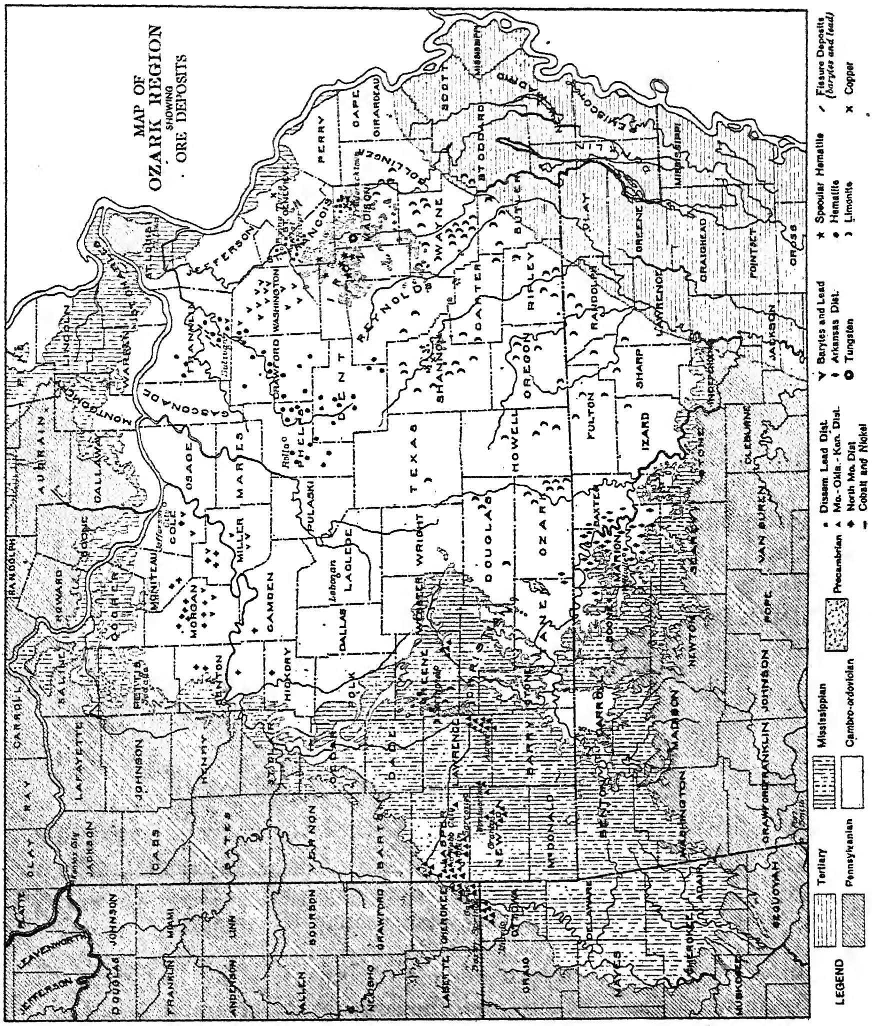



FIG. 1.—COLUMNAR SECTION SHOWING GENERAL DIVISIONS AND FORMATIONS OF THE OZARK REGION AS NAMED IN MISSOURI AND THE VARIOUS HORIZONS AT WHICH THE ORES OCCUR.

The general surface distribution of the larger geologic divisions is shown on the accompanying map (Fig. 2). The location of the various mining districts is indicated by symbol.

The pre-Cambrian rocks outcrop chiefly in Iron, Madison and adjoining counties in southeast Missouri. They consist chiefly of granite and porphyry which have been intruded by comparatively small basic dikes. At Pilot Knob the iron deposits are associated with a porphyry breccia and volcanic tuff which may be of considerably later age. The




granite floor dips away in all directions from the area of outcrop as shown by the following depths at which it has been encountered; Rolla, 1800; Carthage, 1850; Lamar, 1850; Kansas City, 2348; and St. Louis, 3558.

Surrounding the pre-Cambrian area, the Cambro-Ordovician formations occupy the greater part of the Ozark region. They consist chiefly

of cherty and non-cherty dolomites and sandstone. The lower formations, including the LaMotte, Bonne Terre, Davis, Derby, Doe Run, Potosi, and Eminence, outcrop in the area contiguous to the igneous rocks while the Gasconade, Roubidoux, Jefferson City, Cotter, Powell, St. Peter and other Ordovician formations underlie the major portion of the remainder of the region. The Silurian and Devonian occur in a very

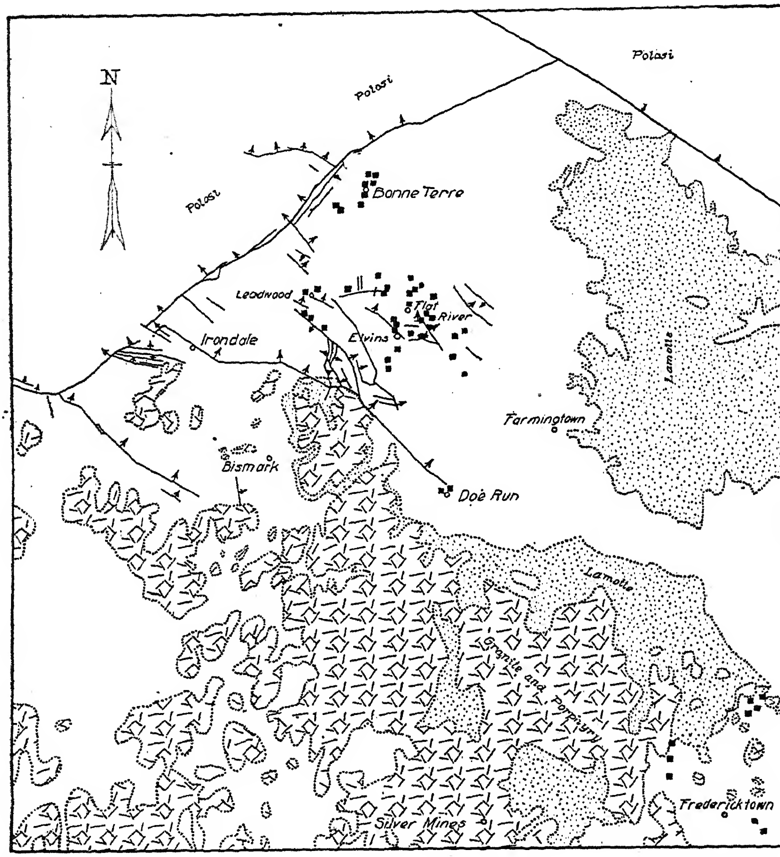



FIG. 3.—SKETCH MAP SHOWING RELATION OF PRODUCTIVE ORE OF THE SOUTHEAST MISSOURI DISSEMINATED LEAD DISTRICT TO FAULTING.

narrow belt along the eastern edge of Missouri. The Devonian also outcrop in northern Arkansas and the southwest portion of Missouri.

The Mississippian overlies the older formations and was deposited originally over most of the Ozark region, but has been removed by erosion and now occupies chiefly the borders of the area.

The Pennsylvanian series overlies the Mississippian and older formations unconformably and practically surrounds the Ozarks along its edges. Like the Mississippian, it was originally deposited over much of the

region but has been removed by erosion and now occurs chiefly as outliers within the region.

Well-marked unconformities indicate the chief breaks in sedimentation, and intraformational unconformities are common in the older formations. The unconformity between the pre-Cambrian igneous rocks and the first sedimentaries was probably of the longest duration. The Mississippian and Pennsylvanian both overlie the older formations unconformably, each series being introduced after an important break in sedimentation.

In general the formations lie approximately horizontal, dipping only slightly away from the central portion of the dome. Along the eastern edge of Missouri, the dip is comparatively steep, and as a consequence the formations outcrop in comparatively narrow belts, almost paralleling the Mississippi River.

The most prominent faulting occurs in the eastern part of the area in the vicinity of the St. François Mountains and in Arkansas where they are associated with the orebodies. Comparatively few faults have been found in the central or western parts of the region and these are more or less local in extent.

Fig. 3 indicates the extent of the fault systems that have been mapped in southeast Missouri. The principal fault zone extending northwest, and southeast along the northern part of the area, has been mapped from near the western boundary of Washington County to Mississippi River, where it crosses into Illinois and evidently is the western extension of the prominent zone passing through the southern part of that State into Kentucky, and further on into Pennsylvania. In Ste. Genevieve County this fault has a throw of at least 2000 ft. (609 m.).

## Ore Deposits

The chief mineral resources of the Ozark region are the ores of lead and zinc, the production exceeding that of any other area in the United States. The major part of the output is obtained from two districts known as the Disseminated Lead district of southeast Missouri, and the Missouri, Oklahoma, and Kansas zinc and lead district, commonly known as the Joplin district, located on the western edge of the region. Minor districts of less commercial value occur throughout the remainder of the area. In north Arkansas, deposits occur over a relatively large area. Throughout the central Ozark region, occasional deposits have been exploited, the major activity having been entered on the northern flank of the uplift. Fissure deposits occur in Franklin County and in Washington County; shallow deposits in the Potosi formation have been worked for many years.

Although less well known, commercial deposits of other metals occur

either associated with the deposits of lead or in separate and distinct areas.

Cobalt, nickel, and copper ores are associated with the disseminated lead deposits in the southern part of that district. Copper has been mined to some extent in Ste. Genevieve and Crawford Counties. Although not exploited, it is found near Eminence in Shannon County at the contact of the sedimentary formations and the pre-Cambrian igneous rocks and in the central Ozark area; copper carbonate is usually encountered in the iron ores mined in that district.

Tungsten is produced in Madison County, where it is associated with argentiferous galena. Various types of iron ore are produced throughout the eastern half of the Ozarks. Pyrite is mined in Crawford and adjoining counties, and barytes is mined extensively in both the eastern and northern parts of the region. The different deposits are easily segregated into districts, as the various types of ore are usually associated with certain formations. The following is a brief description of the various mineral-bearing areas. The individual districts have been described in detail in the publications noted below.[1]

## Disseminated Lead District

The disseminated lead deposits occur mainly in St. François and Madison Counties with comparatively small areas in Washington and Ste. Genevieve Counties. The district has a typical Ozark topography, being for the most part rough and hilly.

The stratigraphic succession is simple, including only those formations up to and including a part of the Potosi as given in the accompanying columnar section.

The pre-Cambrian igneous rocks outcrop in the southern and western portions and underlie the remainder of the area at depths ranging up to 800 or 900 ft. (243 to 274 m.) in the northern part of the district. After the solidification of these rocks, the region was subjected to a long period of erosion, lasting until middle Cambrian time, and the surface was deeply carved by strong drainage systems. With the exception of one

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[1] E. R. Buckley and H. A. Buehler: Geology of the Granby Area. *Missouri Bureau of Geology and Mines* (1905), **4**, 2d ser.

H F. Bain: Preliminary Report on the Lead and Zinc Deposits of the Ozark Region. *U. S. Geological Survey, 22d Annual Report,* Pt. 2 (1901), 23–227.

W. S. T. Smith and C. E. Siebenthal: Joplin District. *U. S. Geological Survey, Folio* 148 (1907).

E R. Buckley: Disseminated Lead Deposits of St. Francois and Washington Counties. *Missouri Bureau of Geology and Mines* (1908), **9**, 2d ser.

C. E. Siebenthal: Origin of the Zinc and Lead Deposits of the Joplin Region. *U. S. Geological Survey, Bulletin* 606 (1915).

G W. Crane: Iron Ores of Missouri *Missouri Bureau of Geology and Mines* (1912), **10**, 2d ser.

or two basic intrusions cutting the sedimentaries in Ste. Genevieve County, there is no evidence of igneous activities after the deposition of the first sedimentary formations. Other than a few gash veins of galena and sphalerite, the igneous rocks are not known to contain lead and zinc ores. Analyses show that these rocks carry traces of both metals.

The LaMotte sandstone was the first sedimentary formation laid down upon the irregular pre-Cambrian surface. It is conglomeratic at the base and has a maximum thickness of from 200 to 300 ft. (60 to 91 m.). It usually shows a transition zone at the top consisting of alternate beds of sandstone and sandy dolomite.

The more elevated portion of the pre-Cambrian area was not covered by the LaMotte sea and the upper formations often rest directly upon the igneous rocks. In the productive district the upper part of the LaMotte is often ore-bearing.

The Bonne Terre dolomite overlies the LaMotte and is the surface formation over a large part of the district. It averages about 350 ft. (106 m.) in thickness and consists of gray to dark non-cherty dolomitic beds. The lower part of the formation is frequently shaley and the basal portion is made up of dolomitic beds having a greenish color, due to the presence of glauconite locally called chlorite. With the exception of the ore occurring in the top of the LaMotte, all of the disseminated deposits are found in the Bonne Terre formation.

The Davis shale overlying the Bonne Terre has a thickness of approximately 160 ft. (48 m.). It consists chiefly of thin bedded dolomitic limestone and shale with several layers of edgewise conglomerate, the latter occurring chiefly in the lower portion. This conglomerate, which is composed largely of small plates of limestone standing on edge, is distinctive of this formation. One hundred feet from the base is a boulder horizon of pure limestone, the boulders being usually less than 6 ft. (1.8 m.) in thickness. This horizon is an excellent geologic marker.

The Derby and Doe Run formations, which overlie the Davis, have an average thickness of 40 and 60 ft. respectively. The Derby consists of heavy bedded, gray dolomite, while the Doe Run is typical "cotton rock" containing small druses of quartz.

The Potosi is the highest formation occurring within the productive district. In the surrounding area it has a thickness of about 300 ft. but within the district is usually represented by chert-covered slopes. The formation consists of heavy bedded, gray dolomite with much interbedded chert, and quartz druses. The latter are typical of this horizon, which is the lowest cherty formation of the Ozark succession.

### *Structure*

In general, the ore-bearing beds in the Bonne Terre-Flat River district have a pitch to the west which is somewhat modified by faulting.

The present productive district is practically outlined by major systems of faults. The fact that the Bonne Terre and lower formations occur at the surface is largely due to this faulting and to the Farmington anticline that brings the LaMotte sandstone to the surface just east of the mining area. Minor faults occur within the area and small faults are frequently encountered in mining, many of which have been subsequent to the deposition of the ore. In a number of instances small faults noted in the lower levels die out and appear as joint planes in the upper workings. Fig. 3 illustrates the relation of the productive area to the principal faulting.

## Ore Deposits

Two types of ore deposits occur within the Bonne Terre formation. Of these the disseminated type is by far the more important, and the only character of deposit being worked at the present time. The early mining, however, was restricted to shallow deposits of massive galena occurring in the surface clay and embedded in clay in joint planes and solution cavities in the upper portion of the formation. These deposits, although rich, seldom extend below 50 ft. (15 m.) and in almost every instance occur above disseminated ore.

The disseminated ore occurs in the Bonne Terre dolomite and to some extent in the upper portion of the LaMotte sandstone. In general, the orebodies occur in large blanket-like deposits varying up to 800 ft. (243 m.) in width and in some instances extending from ½ to 1 mile in length. The ore may be restricted to a single bed or it may occur in different beds so situated that stopes having a height of approximately 100 ft. have been worked. At Bonne Terre, ore has been mined at different levels from near the surface down to the LaMotte sandstone.

The deeper orebodies are of the greatest lateral extent and more uniform in mineralization. The orebodies in the upper part of the formation are irregular and comparatively bunchy, although in many instances very rich.

The ore occurs chiefly in the dark dolomite and in the shales which are carbonaceous. It also occurs abundantly in the "chlorite" zone at the base of the formation. The distribution of the organic material through the formation is very irregular and the general outline of some of the orebodies indicates lagoonlike areas in which organic material was segregated. The light-colored dolomite seldom carries ore.

The major portion of the galena occurs as small crystals disseminated through the rock. It is apparently a metasomatic replacement of the dolomite or shale, thin sections of the latter showing flow lines or compression lines due to the growth of the lead crystal. In the richer deposits, more than 50 per cent. of portions of the original bed has been replaced by galena In part, the ore occurs as crystals lining small

cavities or other openings or is embedded in the clay filling these openings. It also occurs frequently as small crystals along bedding and joint planes and may occur in this manner in the light-colored dolomite where dark beds carrying galena occur above or below.

Well-developed systems of joint planes are encountered in mining. These usually are open and are water-bearing, the rock being oxidized for several feet on either side of the fissure. These fissures usually extend downward to the LaMotte sandstone and their oxidized nature indicates that the surface waters have direct access to the sandstone. Analyses of samples of mine water taken from the upper workings and from the LaMotte sandstone underneath the ore-bearing rocks show a marked similarity, and it appears that the waters of the LaMotte are essentially the same as the solutions near the surface. As a whole, the ore-bearing beds do not show oxidation, but still retain their original dark color. In some instances, however, the beds in the lower part of the formation have been completely oxidized and the galena leached out as shown by numerous casts of the mineral.

In general, the deposits throughout the district show a marked similarity. In the Mine LaMotte area, the clay overlying the shallow deposits carries a notable amount of lead. This clay is the residuum of galena-bearing dolomite from which the carbonates of lime and magnesia have been dissolved.

## Missouri-Oklahoma-Kansas District

This district, commonly known as the "Joplin district," embraces about 3000 square miles, extending from Springfield, Mo., to Miami. Okla. The area is situated on the extreme western edge of the Ozarks, and in general has a gently rolling prairie-like surface, except in the immediate vicinity of the larger streams where the relief is more pronounced.

The first workable ore was discovered at Joplin and Granby, about 1850, and the immediate areas in which the original discoveries were made are still producing. Mining camps have since been developed at different points and the region has experienced a number of mining "booms." However, the discovery of exceedingly rich deposits in Oklahoma and Kansas during the past 2 years has resulted in the most spectacular rush ever experienced in the region. It has in every way equalled the most noted mining booms of the West. At the present time there are approximately 1000 churn drills in operation and nearly 100 mills have been completed or are in process of construction. While the southeast Missouri disseminated lead district makes its production from less than 50 mines operated by five companies, the Missouri-Oklahoma-Kansas district has probably in the neighborhood of 1000 active mines operated by almost as many companies.

## Geology

The surface formations underlying the district are the Cherokee shales of the Pennsylvanian series and the Chester and Boone formations of the Mississippian. The ore deposits occur chiefly in the latter series which constitute the youngest ore-bearing strata of the Ozarks and differ in geologic age very materially from the formations of the disseminated lead district.

Although interrupted by several marked unconformities, the sedimentary succession is comparatively simple, consisting of about 1500 ft. (457 m.) of dolomite and sandstone resting upon the pre-Cambrian igneous rocks. The following generalized section is based upon the record of well No. 6 of the Carthage water-works, Carthage, Mo.

The lowest stratified beds, which rest upon the granite, consist of 200 ft. (60 m.) of sandstone which corresponds to the LaMotte of the southeast Missouri district. Overlying this is approximately 700 ft. (213 m.) of cherty and non-cherty dolomitic limestones which have not been differentiated but which occupy the interval corresponding to the Bonne Terre, Davis, Derby, Doe Run, Eminence, Procter, Potosi, and Gasconade formations of the southeast district. The Roubidoux sandstone overlies these beds, occurring from 800 to 1100 ft. (243 to 335 m.) beneath the surface. Locally it is known as the "sea level" or "water" sand and is the sandstone encountered in the deep wells drilled for water supply. The Jefferson City dolomite overlies the Roubidoux, averaging about 550 ft. (167 m.) in thickness and is the uppermost Ordovician formation of this area.

The Ordovician is separated from the overlying strata by a marked unconformity. Occasionally sandstone is encountered in drilling at this horizon and these remnants may be outliers of St. Peter. They are restricted in area and their exact geological nature has not been determined. Overlying the Jefferson City in the southern part of the district (as far north as the Granby area) a dark shale (Devonian) is encountered. In the Joplin district proper the older formations are overlain by a bluish shale of Mississippian age indicating the absence of the entire Silurian and Devonian periods. The Mississippian shale is the lower portion of the Boone formation over a large part of the district. At the end of Boone time the district was subjected to a long period of erosion and underground solution, resulting in a rather pronounced topographic surface showing sink holes and cave structure; the topography being of the karst type.

During upper Mississippian time Chester formations consisting chiefly of limestones and sandstones were deposited over the area. Erosion prior to the deposition of the Pennsylvanian removed the Chester throughout the major portion of the field except where protected in sink

holes, although in the western portion along Tar Creek these sandstones and limestones are everywhere present.

The Pennsylvanian series represented by the Cherokee sandstone and shale is the youngest formation in the district. Since the deposition of this formation the region has been under continual erosion and to a large extent these strata have been removed from the major portion of the area where it now occurs chiefly as outliers in sinks and other protected points. In the western edge of the field the Cherokee is the surface formation and is almost everywhere present.

The formations have a slight regional dip to the west, the older Ordovician rocks outcropping east of Springfield. Faulting is almost unknown in the district, the Seneca fault, striking northeast and southwest near that city being the only one of prominence.

The ore deposits occur chiefly in the Boone formation and to a lesser extent in the Chester and Cherokee shale. The character of the deposit depends upon the member of the Boone in which it is located.

The upper 100 to 150 ft. (30 to 45 m.) of the Boone is composed of gray crystalline limestone and chert. Underneath this there is a bed of oolite from 2 to 10 ft. thick, known as the Short Creek oolite. While not coextensive with the entire district, the Short Creek is present in all the principal mining districts and forms one of the most valuable geologic markers of the region. Beneath the oolite is 100 ft. of crystalline cherty limestone similar to the upper portion of the formation and from which it can only be distinguished by fossil evidence. The Grand Falls chert locally called "sheet ground" occurs beneath this limestone. It is from 8 to 50 ft. in thickness and is typically developed in the area from Oronogo to Duenweg, and in the west Joplin field. It is not known in the eastern portion of the district at Aurora or near Springfield. The lower portion of the Boone consists of from 50 to 100 ft. of bluish, argillaceous, cherty limestone, known locally as the second lime. It grades below into the shale horizon which is the base of the Boone in the district.

As already mentioned, the Boone after deposition was subjected to a long period of erosion. The coarsely crystalline limestone was easily dissolved and underground water courses were prevalent. Throughout the entire southwestern part of Missouri, large springs issuing from this formation indicate the prevalence of underground streams. The chert which is insoluble has remained as residual material; frequently retaining its bedded character. Sink holes are common throughout the Mississippian area and in the mining region some of these have extended down to the Grand Falls chert member, the sides and bottom being covered with a variable thickness of brecciated chert boulders. The subsequent deposition of the Pennsylvanian has filled these sink holes with shale and sandstone, the general relation being shown by Fig. 4, which also indicates the general relation of the mineral deposits.

Workable orebodies occur in each of the above members. In the upper portion of the formation the ore is found associated with sink-hole structures that are filled with Pennsylvanian shale. The ore usually occurs in the bouldery cherty ground between the limestone formation and the shale. These deposits are very irregular in size and distribution, the dirt being usually very rich. The ground may be soft and require timbering or may be well cemented and the entire orebody worked out with comparatively few pillars. Deposits of this character frequently extend down to the Grand Falls chert and are usually called soft-ground deposits.

The Grand Falls chert member has been rather uniformly mineralized over large areas, the most extensive and typical of which is known as the Webb City-Carterville sheet-ground district.

Where ore-bearing, the Grand Falls chert has been more or less brecciated and the interbedded limestone has been removed by solu-

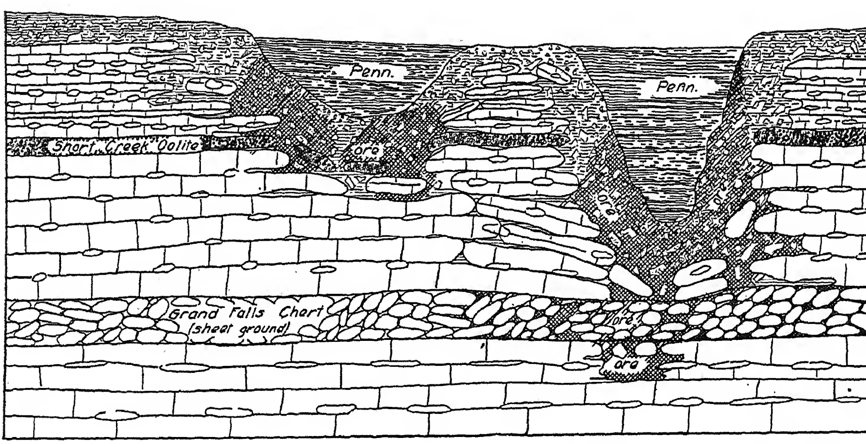


FIG. 4.—CROSS-SECTION SHOWING GENERAL POSITION OF THE OREBODIES OF THE JOPLIN DISTRICT TO THE VARIOUS MEMBERS OF THE BOONE FORMATION.

tion, leaving small cavities and openings. The ore occurs lining these openings or as thin sheets along crevices and bedding planes.

These deposits usually carry a lower percentage of the minerals than the upper soft ground, but their uniform mineralization over large areas has made profitable mining possible through the handling of large tonnages.

In the Miami camp more or less fissuring is noticeable and the orebodies are sometimes sheeted, this structure coinciding with the underground water courses along which the richest ore has been concentrated. At Commerce, a well-defined fault has been encountered in mining, but faults of any consequence are unknown in any of the other camps.

At Sarcoxie, Oronogo, Aurora and Cave Springs, ore has been found

in the cherty limestone underneath the sheet-ground horizon. It occurs through a considerable thickness of the member, as is shown by the high stopes carried in mining, and is usually directly underneath deposits in the overlying members. The shale horizon at the base of the formation carries ore at Springfield and Granby and is mined in the so-called "gumbo runs" of the former area.

In the Miami district the shallow ore deposits occur in the Chester. These usually carry a considerable quantity of bitumen, which is detrimental to mill recovery. The deeper ores of the same area, however, do not contain this material.

The principal ores of the district are sphalerite, calamine, and galena. Minor quantities of smithsonite, cerussite, anglesite, hydrozincite, and leadhillite are found, but these are at present unimportant as ore. The gangue minerals are chiefly chert, dolomite (pink and gray spar), calcite, pyrite, marcasite, limonite, greenockite, chalcopyrite, and quartz. In the shallow workings a soft white to reddish clay known as "tallow" clay is often encountered. This material contains from a trace to 30 per cent. zinc oxide. The calamine is obtained chiefly from the shallow deposits where the subsequent erosion of the carbonaceous Pennsylvanian shales has allowed surface oxidizing waters access to the original sulphide deposits.

There are three recognized varieties of chert associated with the deposits: (1) a dense non-fossiliferous white chert; (2) similar chert that is highly fossiliferous; (3) dark gray to black chert. The white cherts occur in the original Boone formation while the black chert is of secondary origin and has been introduced with the first concentration of the ore. It is cryptocrystalline, gray to black in color and carries crystals of zinc blende and galena when ore-bearing. When oxidized this material turns to a brownish-gray color and is known as "Cod" rock. The black flint is prevalent in most of the mining areas and is associated with the first deposition of the ore.

Where dolomite is prevalent it is designated as "spar" ground, there being both a gray and pink "spar." The gray is the older of the two, apparently being associated with the first introduction of the ore. Where the pink spar is present with the gray, it usually occurs along fissures in the latter, showing its later deposition. The calcite is usually deposited upon the other minerals and is apparently one of the last minerals to be precipitated.

Pyrite and marcasite in some areas are important gangue minerals which materially effect the value of the ore.

In many of the shallow deposits galena predominated, occurring as massive crystals. In the deeper workings zinc blende is the principal ore, the ratio being about 17 of zinc to 1 of lead.

The district at the present time is producing more ore than ever

before and with the completion of the mills now under construction in the Miami and Baxter Springs camps of Oklahoma and Kansas, the output will be materially increased.

## Other Districts

In addition to the two important districts described, there are various minor areas which have distinct types of deposits but which have not proven as important commercially.

To the north and west of the disseminated lead district, chiefly in Washington and Jefferson Counties, the Potosi formation carries widespread deposits of galena associated with barytes. Usually the ore is found in the upper part of the formation along joint planes or horizontal narrow solution channels. The deposits are extremely irregular and no attempt has ever been made to work them on a large scale. Many of the orebodies occur along fault lines and at some points zinc has been found with the lead and barytes. These deposits have been worked since the early days of mining in the region.

In Franklin County near St. Clair, fissure veins have been worked to a depth of 500 ft. (152 m.). These deposits show barytes with lead the galena occupying the center of the vein.

Only a small production is being made by this area at the present time.

In Miller, Morgan, and Moniteau Counties, considerable lead, zinc and barytes has been produced. Much of the lead was recovered from the residual surface clays. The deeper deposits are similar in character to the shallow deposits of the Joplin region, excepting that the country rock is the Jefferson City dolomite rather than the Boone formation. The orebodies are irregular and occur in the brecciated material between the Pennsylvanian shales and the dolomite. In most of the deposits the gangue mineral is barytes and as this mineral cannot be separated from the zinc blende by ordinary milling methods the district has not been developed so as to determine the extent of the orebodies. A number of the Pennsylvanian pockets are composed largely of cannel coal which frequently shows a considerable lead and zinc content. Sporadic deposits of both lead and zinc are found through the central Ozark region.

The most important of the outlying districts is found in the northern part of Arkansas in Marian and adjoining counties. The area is rugged and very hilly, being deeply eroded by White River and its tributaries. While orebodies occur in the Boone, the chief deposits are worked in the Cotter and Powell formations, formerly known as the Yellville. Smithsonite (carbonate of zinc), calamine (silicate of zinc) and sphalerite are the chief minerals of zinc, while galena and cerussite constitute the main lead minerals.

Faulting is more prevalent in this district than in any other except

southeast Missouri, and many deposits occur along these fault planes. The principal deposits outcrop on the hillside and consist of oxidized ore. The orebodies usually follow underground solution channels.

### Cobalt, Nickel, and Copper

Associated with the lead deposits of the southern portion of the "disseminated lead" district of southeast Missouri are extensive deposits of cobalt, nickel, copper, iron, sulphides, constituting a complex ore difficult to reduce. The known deposits occur in the vicinity of Fredericktown and these ores are now being produced by the Mine LaMotte Co. and Missouri Cobalt Co.

The ore occurs chiefly in the LaMotte sandstone and the lower portion of the Bonne Terre dolomite near the contact with the underlying pre-Cambrian formations. It assays from 1 to 2 per cent. cobalt and nickel and about 2 per cent. copper.

In the Flat River-Bonne Terre area but little of this character of ore occurs, although the middling product of some of the tables separates mixed sulphides.

Copper ore, chiefly carbonate, has been found in Shannon County at the contact of the pre-Cambrian and sedimentary rocks. In the Slater shaft west of Eminence, a small stringer of the ore has been followed to over 100 ft. in depth where the joint plain has been reduced to less than an inch between walls. In many cases the deposits of hematite in the filled sink area also show copper carbonate but in no instance has copper ore been produced on a commercial scale.

The orebody of the Copper Mountain Copper Co. southeast of Sullivan has shown some sulphides near water level in addition to considerable carbonate. In Ste. Genevieve County, copper ore has been mined from cherty brecciated beds of the Cotter-Powell formations. These deposits occur chiefly along intraformational conglomerates and are bunchy and irregular. Other than the ore produced at Fredericktown, the Ste. Genevieve area is the only one that has produced commercially.

### Tungsten Deposits

Tungsten is being mined in the granitic area of Madison County, Missouri, about 12 miles west of Fredericktown. The property was first opened in 1877 and during a short period was operated for the recovery of argentiferous galena. Recently it was re-opened for the recovery of tungsten which is associated with the galena.

The deposits occur in quartz fissure veins carrying galena and tungsten ore. At the present time, seven distinct veins have been opened. In general, they dip approximately 60° to the southeast and

have the average width of about 2 ft. Huebnerite, which occurs scattered through the quartz and in massive bunches, is a main ore of tungsten. Iron pyrites and dark mica are usually associated with the ore and the galena found chiefly in the clay gangue has been reported to run as high as 40 to 60 oz. of silver per ton. This is the only occurrence of argentiferous galena and tungsten known in the Ozark region.

## Iron Ores

Although at the present time the iron industry is of small commercial value, formerly Missouri was one of the chief iron-producing sections of the United States. Several types of commercial ore have been mined, each of which is so segregated as to form a distinct district. The chief types have been designated as follows:

### *Specular Ore in Porphyry*

These deposits occur chiefly at Iron Mountain and Pilot Knob, where the ore is associated with the pre-Cambrian porphyry. Although not being extensively operated at the present time, they have produced approximately 10,000,000 tons of high-grade hematite.

### *Hematites of the Filled Sinks*

The red hematite deposits occur chiefly in Crawford and adjoining counties in Missouri. The ore occupies old cave or sink structures that were originally filled with marcasite and pyrite. Through oxidation, these sulphides were later altered to a high-grade limestone. The ore-bodies vary in size and may contain from a few thousand to a million tons, the Cherryvale Valley mines having produced the latter figure. The ore usually outcrops or is found near the surface and a majority of the properties have been worked by stripping.

### *Brown Ores*

Both primary and secondary limonites occur chiefly in the eastern Ozark area of Missouri and Arkansas, although minor deposits occur in the western portion of the area.

Both types of ore are found in the surface clay.

The secondary ore is chiefly a stalactitic ore having a pipe-like structure. It was originally deposited as sulphide but has been completely oxidized to limonite. The primary ore is frequently cherty but shows no pseudomorphs after sulphides.

## Barytes

Missouri is one of the chief producers of barytes in the United States and until the past 2 years averaged more than 50 per cent. of the output of the country. About 75 per cent. of the production comes from Washington County, where the mineral is found, largely embedded in the surface clay. It is associated with more or less lead ore and is frequently highly colored by iron oxide. In Jefferson County barytes occurs in the fissure veins as a gangue. In the northern Ozark area it is also found associated with both lead and zinc ores.

## Genesis

The lead and zinc deposits of the Ozark region have been attributed by various investigators to: (1) deposition by ascending solutions rising from unknown depths; (2) original deposition in the formation in which the ore is now found and concentration by lateral secretion; (3) deposition by ascending and enrichment by descending solutions; and (4) deposition by descending solutions.

With the exception of the specular hematite of the porphyry region and the quartz veins carrying tungsten and argentiferous galena in Madison County, both of which are probably deposited from hot solutions rising from unknown depths, the author believes the various ores of the Ozark region to be the result of concentration by descending solutions at ordinary temperatures. There has also been important lateral movement of the waters along fissures and solution channels.

The presence of large springs throughout the region indicates a well developed underground drainage, and the occurrence of numerous sink holes and large caves show the extensive solution of the original limestones and dolomites. Not only has this action been prevalent during the present erosion period, but similar conditions occurred at the end of Boone and Chester time.

There is no evidence throughout the region that the sedimentary formations were ever subjected to the action of hot solutions or gaseous emanations. They are in no way metamorphosed so as to indicate action of heat. The sandstones are not changed to quartzite but are in the main friable and loosely cemented. The dolomites are granular to crystalline and in every respect indicate normal shallow-water deposits. The shaley horizons do not carry silicate minerals such as might indicate the action of deep-seated waters.

It is thought that the minerals were first deposited in minute particles through the various sedimentary formations, the original source being the pre-Cambrian igneous rocks, subsequent action by weathering and descending solutions having resulted finally in the concentration of the

ores in their present position. Analyses have shown that lead and zinc occur in minute quantities in all of the older formations underlying the Ozarks.

With the exception of the disseminated lead district, the various ore deposits are associated with open structures formed, at least in part, prior to the introduction of the ore-bearing solutions. These openings are largely solution cavities formed by descending waters prior to the deposition of the Pennsylvanian.

The formations of the productive part of the disseminated lead district have been elevated by faulting and subsequent erosion has cut through the overlying formations down to the Bonne Terre dolomite which is the surface formation over a large part of the area. Well developed joints extend from the surface to the underlying LaMotte, the oxidized condition of the rock indicating that surface solutions have direct access to the sandstone. Analyses of the waters obtained from the sandstone and from the upper part of the mine workings are practically identical. In the productive area the Davis shale, which would prove the limiting factor in either an ascending or descending circulation, has been cut through by erosion allowing waters to circulate downward, while in the area beyond the main faults this shale and succeeding formations overlie the Bonne Terre continuously, limiting any downward circulation. In the latter area disseminated deposits have not as yet been found.

Shallow deposits of massive galena occurred in the upper part of the Bonne Terre near the eroded edge of the Davis shale. These deposits, which in the aggregate contained a fairly large tonnage of lead, are undoubtedly due to descending solutions and are a strong argument that descending surface solutions of this district have carried the metal.

It is probable that solutions from the LaMotte sandstone in part circulate upward at least through the lower portion of the Bonne Terre. These solutions are, however, essentially surface waters which have entered the sandstone directly through fissures in the Bonne Terre or through the large outcrop area east of Farmington where the Farmington Anticline has brought the LaMotte to the surface.

The shallow deposits of galena and barytes occurring in the Potosi formation in Washington and Jefferson Counties are evidently the result of descending solutions. The ore occurs in irregular solution channels near the top of the formation. The Davis shale underlies the Derby, Doe Run and Potosi formations which occur continuously throughout this region. It has a normal thickness of over 150 ft. (45 m.) and forms an impervious horizon through which ascending waters cannot penetrate.

The iron ores of the central and southeast Ozark region occur at or near the surface, the original deposits being in solution cavities.

The secondary limonite deposits consist largely of stalactitic ore

known locally as "pipe ore." It is composed of small stalactites originally deposited as sulphides (marcasite and pyrite) in small openings in the once overlying dolomites and sandstones and the ores are now found embedded in the surface clays, due to the removal of the limestone by subsequent erosion. During the erosion period, the sulphides were oxidized to limonite. Although the ore is not found in place, the stalactitic character of a large percentage of the deposits can only be accounted for by descending solutions.

The hematite ores of the north-central Ozarks occupy sink or cave structures throughout an area formerly overlain by the Pennsylvanian. The Roubidoux and Gasconade formations in which the deposits occur do not show depletion of iron content in the vicinity of the orebodies, in fact, these formations carry only a small amount of iron. The Pennsylvanian, however, was rich in iron and it is thought to have been the source of the ore, which was first deposited as the sulphide. As the carbonaceous shales of the Pennsylvanian were gradually removed by erosion, the orebodies were more subjected to the action of surface waters which oxidized the sulphides, leaving the deposits in their present condition.

There is no evidence in any of the deposits that the concentration was due to ascending waters. Along the sides of many of the orebodies, boulders of the country rock are embedded in the ore, indicating solution during the time of deposition. It is believed that any ascending solutions would be so saturated with lime and magnesia while passing through the lower formations that they would have no solvent power upon reaching the ore horizon. Descending waters, on the other hand, have a strong solvent action.

In the Missouri-Oklahoma-Kansas district, the orebodies, as already mentioned, occur in sink structures and solution channels, or are associated with brecciation that is in large part the result of solution. These openings are evidently the result of descending waters and are confined to the rocks above the shale occurring at the base of the Mississippian.

It is believed that the ores were concentrated after the deposition of the Pennsylvanian by descending solutions. These deductions are based in part upon the following general field evidence.

1. In places the sheet ground shows brecciation and cementation by silica without the introduction of ore or organic material. Later this breccia has been rebroken and recemented with black secondary flint-carrying ore. It is evident that the first circulation occurred prior to the circulation carrying metals in solution, and indicates that during the early periods of erosion (pre-Pennsylvanian) the solutions did not carry ore-forming constituents. If these solutions were ascending they should have contained the same mineral-forming constituents as in any subsequent ascending circulation, and for this reason the first cementation should show ore and organic material.

2. The gangue minerals vary in different areas, some mines showing a preponderance of dolomite while others show chiefly black flint. The amount of iron sulphide, chalcopyrite, greenockite, quartz, and other gangue minerals varies materially throughout different mines often in the same camp. The same is true of the relative amounts of lead and zinc. Descending solutions gathering their materials from different sources would naturally result in deposits differing in composition, while ascending solutions which have passed for long distances through dolomite should be uniform in composition and result in uniform deposit. Such solutions would be saturated with lime and magnesia and have practically no solvent powers at the end of their journey, yet the deposits of the district show extensive solution during the time of deposition.

3. The organic matter occurring in the black chert has evidently been derived from the overlying black Pennsylvanian shales and not from formations underneath, as these contain but little organic material and do not show any similar concentration of black flint at any horizon.

4. Secondary enrichment cannot be determined in the area. The extensive oxidization of the upper portion of many of the orebodies shows the action of descending solutions at present, and if these solutions are now carrying lead and zinc downward and reprecipitating them on ores originally deposited by an ascending solution, there should be a marked difference in the character of the earlier and later deposits.

5. The various camps do not show an artesian circulation. In general, deep wells drilled to the Roubidoux sandstone do not indicate artesian conditions until the sandstone is encountered. The Jefferson City dolomite, underlying the Mississippian, is approximately 500 ft. thick and is seldom water-bearing. When the head is pumped off in any mining camp the drainage is only such as would be expected under ordinary conditions.

6. The productive area is not faulted nor does it show fracture zones beneath the ore-bearing horizon that would give free access to ascending circulation. The shaley lower Burlington limestone and the Chattanooga and Mississippian shales which underlie the area are effective barriers to ascending waters.

# The Ferrous Iron Content and Magnetic Susceptibility of Some Artificial and Natural Oxides of Iron

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(St. Louis Meeting, October, 1917)

## Introduction

It is well known that ferric oxide, $Fe_2O_3$, is paramagnetic, while magnetite, $Fe_3O_4$, is classed among the highly ferromagnetic substances. But magnetic data on oxides intermediate in composition between $Fe_2O_3$ and $Fe_3O_4$ have been almost completely lacking.

$Fe_2O_3$ and $Fe_3O_4$ form a solid-solution series, according to present evidence.[1] In this series the properties change continuously from $Fe_2O_3$ toward $Fe_3O_4$, as the percentage of FeO increases from zero toward 31.03, which is the percentage in magnetite. It is possible that there is a break in the series near $Fe_3O_4$, but it has not yet been possible to establish the existence of such a break experimentally.

The occurrence of solid solution in this series is shown by the dissociation pressure, or oxygen pressure in equilibrium with the oxides, which falls continuously over the range from $Fe_2O_3$ to $Fe_3O_4$. The oxygen-pressure curves at 1100° and 1200° are shown in Fig. 1. The existence of solid solution is also demonstrated by the continuous change in optical properties from $Fe_2O_3$ over to a composition containing about 18 per cent. FeO, at which point the opacity of the oxide becomes so great that it is impossible to obtain measurements at higher percentages of ferrous iron.

Natural oxides of iron intermediate in composition between $Fe_2O_3$ and $Fe_3O_4$ are much more common than is generally supposed. If a mineral oxide is not strongly attracted by a small hand magnet and if it gives a red streak, it is usually labeled "hematite." If it is strongly attracted by the hand magnet it is usually labeled "magnetite" without further tests. As we shall show later, oxides containing from 1 up to 31 per cent. FeO can be thus erroneously lumped together as "magnetite." It is a fact, nevertheless, that the great bulk of natural oxides of iron lie fairly near either $Fe_2O_3$ or $Fe_3O_4$ in composition. The reason will be

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[1] R. B. Sosman and J. C. Hostetter: The Oxides of Iron, *Journal of the American Chemical Society* (1916), **38**, 807–833.

evident from the shape of the dissociation-pressure curves of Fig. 1. If drawn to a direct scale instead of a logarithmic scale of pressures, the middle portion of the curves would be decidedly flat, with a rapid rise near $Fe_2O_3$ and a rapid fall near $Fe_3O_4$. It is clear, therefore, that at a given temperature only a certain narrow range of oxygen pressures is available for the production of oxides of say 2 to 29 per cent. FeO, while all other pressures would produce oxides approaching pure $Fe_2O_3$ or pure

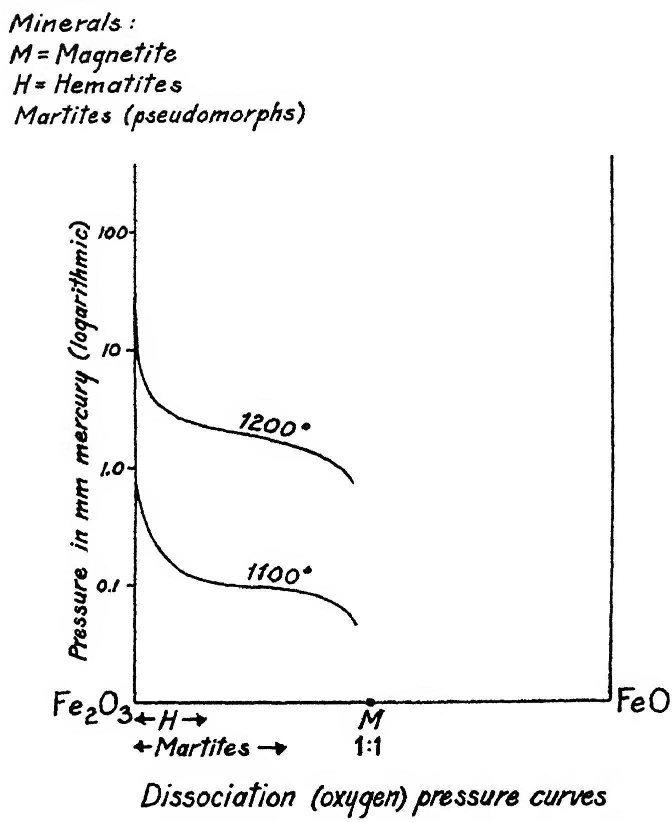


FIG. 1.—OXYGEN PRESSURE CURVES IN THE SYSTEM $Fe_2O_3$–$Fe_3O_4$.

$Fe_3O_4$ in composition. The solid-solution series of the oxides of iron thus differs from the solid-solution series of the lime-soda feldspars in that the probability of occurrence of any given member of the feldspar series is much more independent of the composition than is the probability of occurrence of any given member of the iron oxide series.

Most of the hitherto published analyses of iron ores and iron oxide minerals show the total iron, but not the amount of iron which is in the ferrous condition. We find that practically all natural oxides of iron contain a determinable amount of FeO. Specular and micaceous hema-

tites, especially, show notable percentages of ferrous iron, and at the same time are often markedly magnetic. The experiments reported on in this paper were undertaken in order to find what relation exists between the ferrous iron percentage and the magnetic properties.

That there is some relation between the two was recognized as long ago as 1848 by Plücker,[2] who observed that some hematites were much more magnetic than others. He was of the opinion that the magnetism is due to small percentages of FeO, which he thought might be determined quantitatively by measuring the susceptibility of the oxides.

The term "magnetic" as applied to a mineral usually means little more than that the mineral is noticeably attracted by a small horseshoe or bar magnet. The impression gained from such tests is that the magnetic susceptibility increases much more rapidly than in proportion to the percentage of FeO, since a hematite with only a fraction of 1 per cent. FeO can be picked up by a hand magnet just as magnetite itself can be picked up. But the field of such a magnet is so irregular, and so intense close to the iron, that it gives no adequate quantitative idea. Measurements presented in this paper show, in fact, that the magnetic susceptibility seldom exceeds a value proportional to the percentage of FeO.

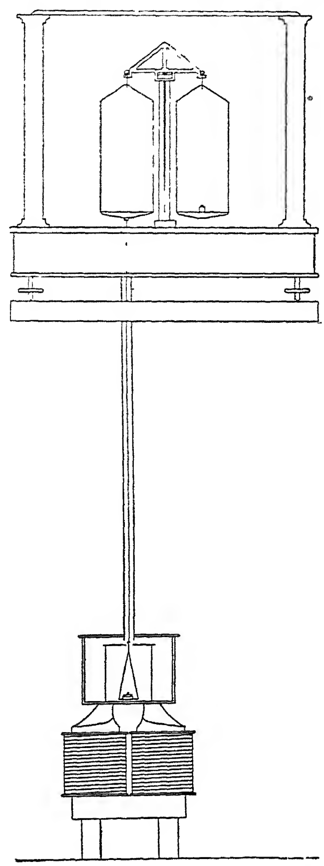
Fig. 2.—Apparatus for comparative magnetic tests.

### Apparatus and Method

The comparative magnetic susceptibility of the natural and artificial oxides of iron was determined by weighing the force exerted upon a known weight of the substance by a non-uniform magnetic field.

The apparatus used for the magnetic tests, as shown in Fig. 2, consisted simply of a Rueprecht analytical balance, from one pan of which was suspended a glass platform on which the charge of oxide was placed, a short distance above the poles of an electromagnet. The oxide was contained in a small flat-bottomed glass pan, and the particles were kept from moving by a glass plunger which pressed down the powder. The steel

[2] Plücker: Über Intensitätsbestimmung der magnetischen und diamagnetischen Kräfte. *Poggendorff's Annalen* (1848), **74**, 321–379.

pointer of the balance was replaced by a pointer made of brass; the polarization of the steel pointer would otherwise have caused an appreciable deflection, necessitating a large correction to the results. The steel knife edges caused only a small deflection, easily corrected for. Blank measurements made with no oxide in the glass pan showed a maximum correction of 0.11 mg. One gram of crystallized powdered ferrous ammonium sulphate, at the maximum magnetizing current, was attracted by a force of 2.68 mg., which could be determined within 0.03 mg.

The distance of the glass platform above the poles was 5.1 mm. A variation of 1 mm. in this distance was found to produce a change of 0.22 mg. in the attraction, in a total of 6.02 mg. The error due to movement of the balance or electromagnet was therefore inappreciable, as such movements were found to be of the order of 0.1 mm.

This simple form of apparatus serves very well for comparative tests, and it is possible to obtain with it data on the true susceptibility of paramagnetic substances, in which the susceptibility does not vary appreciably with the field strength. But the field is rather concentrated, and it is difficult to determine what is the value of the magnetizing force at a given point. The iron core of the electromagnet introduces the further complications that the magnetizing force is not only a complex function of the current, but also is different according as the current is being increased or decreased, on account of the hysteresis of the core. As we wished to obtain comparative data not only on susceptibility but also on the relation between magnetizing force and induction, as well as on the remanence and coercive force, of the entire series of oxides to be examined, the electromagnet was later replaced by a set of solenoids of known constants, and other improvements were made in the apparatus at the same time. The new solenoid apparatus is now being used for further experiments, but the data given in this paper were all obtained with the simpler electromagnet form.

*Analysis of Samples*

The ferrous iron was determined by solution in hydrochloric acid in an atmosphere of carbon dioxide, and titration with potassium permanganate. The method has been described in detail in a previous publication.[3]

Percentages of FeO of the order .of 0.1 per cent. or less cannot be very accurately determined by titration with permanganate in the presence of chlorides, even with the addition of the usual "titrating mixture," as the quantity of permanganate solution required is not much more than that necessary to give the end-point color. Samples containing very little FeO were therefore titrated electrometrically, by measurement of the oxidation-reduction potential of the solution, using a platinum elec-

[3] J. C. Hostetter: *Journal of the merican Chemical Society* (1916), **38**, 821.

trode against a tenth-normal calomel electrode, with dilute potassium dichromate as the titrating solution. A few thousandths of a per cent. of ferrous iron can be detected and estimated by this means. Description and tests of the method will be presented in a later publication.

As will be evident from the results presented later in Tables 5 and 6, the permanganate method, though it gives only an approximate value when the percentage of FeO is small, agrees very well with the electrometric dichromate method in the cases where both were tried on the same sample. The difference is large only in the case of No. 1,521, on which electrometric gave 0.275 per cent., permanganate 0.15 per cent. The error in the permanganate titration in this case must have been accidental. The very small quantity of sample available did not permit of a repetition of the analysis.

### *The Ferromagnetic Members of the Series*

For paramagnetic substances, such as ferrous ammonium sulphate and ferric oxide, the weight and shape of the charge and its distribution in the magnetic field have very little effect upon the magnetic attraction. But as the susceptibility rises in the series and the oxides become more ferromagnetic, the attraction is affected by several variables besides the amount of ferrous iron in the sample. It is a characteristic of the so-called ferromagnetic substances that the magnetization in a given field varies with the shape of the fragment, with the fineness of grain (if the substance be powdered), with the distribution of the material in the magnetic field, and with the nearness of the particles to each other. In addition, the particles often possess a certain remanence, that is, become permanently polarized, so that their attraction in a given field also depends upon their previous magnetic history. The weighing of the magnetic attraction, therefore, seems to the physicist a very crude means of getting at the magnetic properties of a ferromagnetic material, properties which are perfectly definite and measurable by familiar methods, provided only that a large and homogeneous sample can be obtained. But large size and homogeneity are just the characteristics which are most conspicuous by their absence in the case of natural minerals, and we must therefore be content with a method which permits the use of very small quantities of powdered materials.

By powdering to about the same degree of fineness and distributing the samples similarly, good comparative results can be obtained, as will be shown in the following section.

## Tests on Magnetite

For comparison with the results on natural and artificial hematites, and to test the reliability of the method, a series of measurements was

made on magnetite and magnetite-quartz mixtures. Since magnetite is the most ferromagnetic member of the $Fe_2O_3$-$Fe_3O_4$ series, it is to be expected that the variations in comparative results obtained by changing the variables will be at a maximum in the case of magnetite, and will be less for other members of the series in proportion to their smaller magnetic susceptibility.

Crystals of magnetite from the Barton Hill mine at Mineville, N. Y. (sample 3*b*), were crushed and sized through Tyler Standard sieves. The portion which passed 325 mesh (opening 0.038 mm.) was further fractionated by means of an air jet, in an air elutriator similar to that used by the Bureau of Standards for the classification of cement,[4] but of smaller size.

In the earlier tests, the magnetizing currents used were 110, 400, 800, and 400 milliamp., in the order named. In order to obtain a better indication of the shape of the magnetization and hysteresis curves of the specimens, later tests were made with currents of 515, 800, and 485 milliamp., since it was found experimentally that ferrous ammonium sulphate was attracted with just one-half the maximum force (due to 800 milliamp.) by a rising current of 515 or a falling current of 485 milliamp. For the present we are interested only in the comparative attractions produced by the maximum magnetizing current of 800 milliamp.

The reproducibility of the measurements was tested on 500-mg. portions of the magnetite powder between 170 and 200 mesh (sieve openings 0.089 to 0.074 mm.). The results are shown in Table 1.

TABLE 1.—*Reproducibility*

| | Mg per Gram |
|---|---|
| New sample | 9,330 |
| Same, taken out and thoroughly mixed | 9,258 |
| Same, taken out and thoroughly mixed | 9,261 |
| New sample | 9,282 |
| Mean | 9,283 |

The reproducibility of the measurements is thus well within 1 per cent.

The effect of the weight of the charge upon the attraction per gram was tested on the same size powder (170 to 200 mesh) with the result shown in Table 2.

TABLE 2.—*Effect of Size of Charge*

| Mg | Mg per Gram |
|---|---|
| 500 | 9,283 |
| 350 | 9,395 |
| 100 | 9,549 |

[4] J. C. Pearson and W. H. Sligh: An Air Analyzer for Determining the Fineness of Cement. *U. S. Bureau of Standards, Technologic Paper* No. 48 (1915).

For a fivefold increase in the quantity of oxide tested, there is an increase of about 3 per cent. in the attraction per gram.

The size of the particles has a considerable effect, as shown by the figures in Table 3, obtained on 500-mg. charges.

TABLE 3.—*Effect of Size of Particles*

| Size | Diam Particles, Mm. | Attraction Mg per Gram | Per Cent. FeO | Per Cent Insoluble Residue |
|---|---|---|---|---|
| 100–115 mesh........... . . | About 0 16 | 9,516 | 29.66 | 2.62 |
| 170–200 mesh............. ... | About 0.12 | 9,283 | | |
| 300–325 mesh........... . .. | About 0 07 | 8,982 | | |
| 2-mm. air jet ............... | 0 05 and smaller | 8,718 | 29.62 | 2.71 |

Before the effect found can be ascribed to difference in size of the particles, it must first be shown that the composition of the finest material is not markedly different from that of the coarsest material. Not only might the finest powder be expected to be more oxidized than the coarse, but the insoluble quartzitic residue might be expected to be unequally distributed among the fractions. The results shown in the last two columns of Table 3 were obtained by solution in HCl under a carbon dioxide atmosphere followed by electrometric titration. The figures show that the coarsest and finest fractions are practically identical in composition. The difference in magnetic attraction, amounting to about 9 per cent., must therefore be ascribed to the difference in size of the particles.

Taken as a whole, the tests show that satisfactory comparative results can be obtained by the method of weighing the attraction in a strong field, even on a ferromagnetic oxide, and that if desired the effect of certain variables can be evaluated. But for comparison over a range of 1 to 5000, which is approximately the range in the hematite-magnetite series, even variations such as those due to different sizes of particles are of secondary importance, and comparisons can be made without carefully sizing the powder, which might be difficult to do on such small quantities as were available from some of the samples.

TABLE 4.—*Effect of Dilution of Magnetite*

| | | | | |
|---|---|---|---|---|
| Percentage of original magnetite in mixture............ | 100 | 50 | 10 | 1.0 |
| Magnetic attraction, mg. per gram of mixture: | | | | |
| 1. If directly proportional to percentage............. | 8,718 | 4,359 | 872 | 87.2 |
| 2. If proportional to percentage, but corrected for varying amount of magnetite present (Table 2)..... | 8,718 | 4,441 | 900 | 90.3 |
| 3. As found experimentally........................ | 8,718 | 3,896 | 735 | 69.7 |

The effect of dilution of the magnetite was tested by mixing weighed

portions of the 2-mm. jet size magnetite with powdered quartz which had been ground to pass 200 mesh (0.074 mm. and finer). The results are shown in Table 4.

These results show that dilution of the magnetite diminishes the induction by an amount which is greater, the greater the dilution. This is to be expected, because placing the individual particles

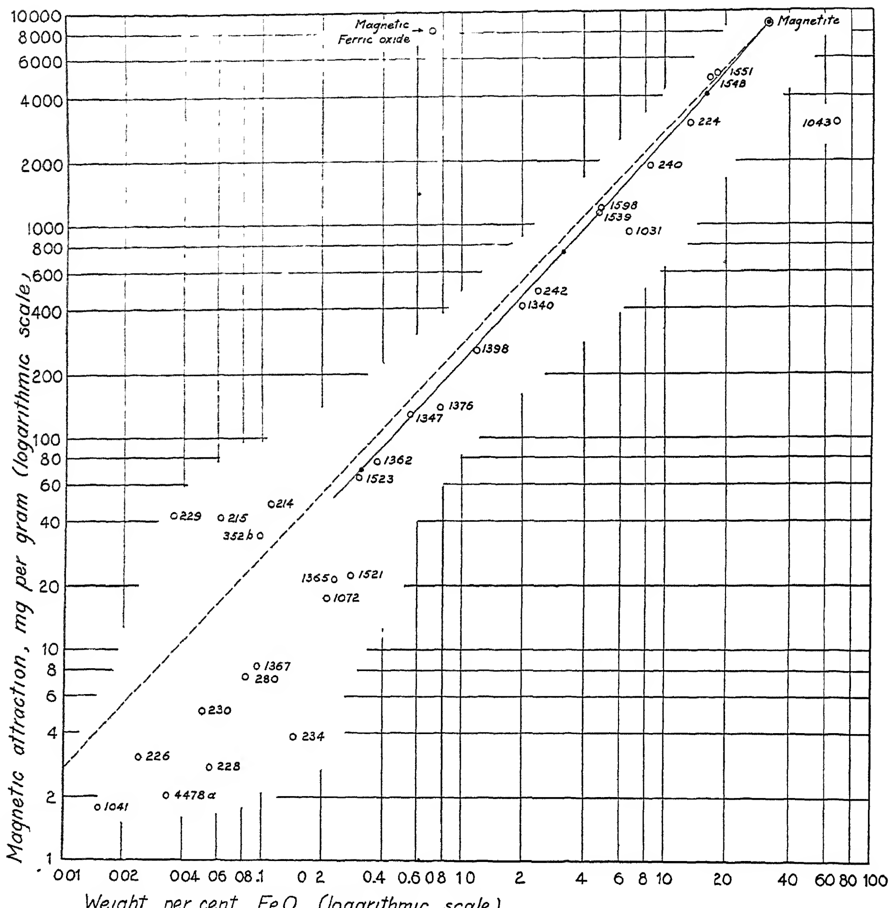


FIG. 3—RELATION OF MAGNETIC ATTRACTION TO PERCENTAGE OF FeO, ARTIFICIAL OXIDES.

farther apart should diminish their mutual inductive effect. The results are shown graphically by the solid curve and the black dots in Fig. 3. The broken curve in the same figure represents the relation that would exist between magnetic attraction and percentage of ferrous iron if the attraction were strictly proportional to the amount of magnetite present.

The existing data on the dilution of magnetic powders are not very complete and are also not in very good agreement. Auerbach[5] used very finely divided chemically reduced iron powder, and diluted it with wood powder. The density of the iron in the mass was varied from 0.0176 to 3.612. Within this range the magnetization increased with the density, not linearly nor uniformly, but according to a curve which is expressible as a power series beginning with a term which expresses the magnetization at infinite dilution.

Benedicks[6] diluted magnetite (size between 0.4 and 0.2 mm. sieves) with powdered quartz and obtained a curve of form somewhat different from Auerbach's, but agreeing with Auerbach in that the magnetization increased with the concentration of the magnetite. Iron-copper and nickel-copper powdered mixtures gave similar curves. Benedicks believes that the peculiar form of Auerbach's curve resulted from some unrecognized variable.[7]

Our curve, being in terms of magnetic attraction per gram and concentration by weight, is not directly comparable with the results quoted above, which are in terms of intensity of magnetization and concentration by volume. Our curve agrees with those in showing that the effect of dilution is to diminish the magnetization. The effect is less than that indicated by Benedicks' curves. This may be due to the fact that our field was much more intense than the field of 143 gauss used by Benedicks; as Maurain[8] has shown, the magnetization tends to become more nearly independent of the concentration as the field becomes stronger.

### *Oxidation of Magnetite by Grinding*

The following results on a mixture of quartz and magnetite suggested that $Fe_3O_4$ could be oxidized to paramagnetic $Fe_2O_3$ by grinding. The original mixture contained 0.635 per cent. $Fe_3O_4$:

| | Magnetic Attraction at 600 Milliamperes, Mg per Gram |
|---|---|
| Original mixture | 26.68 |
| Ground 15 min | 22.58 |
| Ground 15 min | 20.45 |
| Ground 15 min | 19.08 |

It was not certain from these tests whether the change was due to increased fineness, or whether oxidation was partly responsible. Analyses

[5] F. Auerbach: Magnetische Untersuchungen. *Wiedemann's Annalen* (1880), **11**, 353–394.

[6] C. Benedicks: Experiments on Allotropy of Iron: Behaviour of Ferromagnetic Mixtures; Dilatation of Pure Iron. *Journal of the Iron and Steel Institute* (No. I, 1914), **89**, 407–459.

[7] *Op. cit.*, 414.

[8] C. Maurain: Sur les propriétés magnétiques des poudres de fer. *L'Éclairage électrique* (1903), **34**, 465–472.

TABLE 5.—*Ferrous Iron Content and Comparative Susceptibility of Artificial Oxides of Iron in Powder Form*

| Specimen No. | Magnetic Test No. | Designation of Specimen | Source | Color of Powder | | Wt. of Sample Tested, Mg. | Temp | Per Cent. FeO | Method* | Magnetic Attraction per Gram at 800 Milliamp. | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | No. | Name | | | | | In Mg. | Relative |
| 6,211 | 1,634<br>1,638<br>1,645<br>1,600<br>1,801 | Ferrous ammonium sulphate, $FeSO_4.(NH_4)_2SO_4.6H_2O$ | J. T. Baker Chem. Co. (Lot 6,211) | ......<br>......<br>......<br>......<br>...... | ......<br>......<br>......<br>......<br>...... | 970.7<br>1,004.<br>935.7<br>1,055.4<br>1,158.1 | ±25.<br>21.6<br>23.2<br>±22.<br>±23. | ......<br>......<br>......<br>......<br>...... | ......<br>......<br>......<br>......<br>...... | 2.72<br>2.71<br>2.71<br>2.63<br>2 61 | |
| | | | | | | | | | | 2 68 | 1 00 |
| 4,913 | 1,632 | Ferrous sulphate, $FeSO_4\ 7H_2O$ | Do. "Special." (Lot 4,913) | ...... | ...... | 1,242.8 | 27. | ...... | ...... | 3.51 | 1.31 |
| 1,041 | 1,635 | Reagent iron oxide | Merck (Lot 1,041) | 5″ *l* | Ocher red to Prussian red | 1,014. | 22.2 | 0.015 | *E* | 1.77 | 0.66 |
| 4,478*a* | 1,637 | Eisenoxyd zur Analyse mit Garantieschein | Kahlbaum (Lot 4,478) | 65‴ *l* | Raisin black to dark slate-purple | 1,020. | 22. | 0.033‡ | *E* | 2 01 | 0.75 |
| 228 | 1,676 | Ferric oxide, ignited over blast lamp | Ferric nitrate, J. T. Baker | 69‴ *l* | Anthracene purple to taupe brown | 1,107.0 | ±20. | 0.055 | *E* | 2.76 | 1.03 |
| 226 | 1,674 | Ferric oxide, ignited over blast lamp | "Ferric hydrate, C P. moist," J. T. Baker, 72,612 | 69‴ *m* | Taupe brown | 1,024.0 | ±21. | 0.024 | *E* | 3.02 | 1.13 |
| 234 | 1,681 | Ferric oxide, ignited over blast lamp | Ferric nitrate, 2*a*, from electrolytic iron | 65‴ *l* | Dark slate-purple to raisin black | 1,039 9 | ±21. | 0.143 | *E* | 3.82 | 1.43 |
| 230 | 1,677 | Ferric oxide, ignited over blast lamp | Ferrous sulphate, J. T. Baker | 65⁗ *k* | Dark heliotrope slate | 1,049.6 | ±21. | 0.050 | *E* | 5 08 | 1 89 |
| 268 | 1,837 | Ferric oxide, heated in oxygen at 1,100° | Ferric nitrate 2*a*, from electrolytic iron | 49′′′′′ *j* | Dark to blackish plumbeous | 1,064.3 | ±23. | ...... | ...... | 6.92 | 2 58 |
| 274 | 1,837 | Same heated in air at 1,000° | ...... | 49′′′′′ *j* | Do. | 957.4 | 23. | ...... | ...... | 7.13 | 2 66 |
| 280 | 1,838 | Same heated in oxygen at 1,000°–1,100° | ...... | 49′′′′′ *k* | Blackish plumbeous | 1,009.0 | ±23. | 0.082 | *E* | 7.39 | 2.76 |
| 231 | 1,678 | Ferric oxide, dried at 110° | Hydrolysis solution Kahlbaum nitrate, 200° | 5′ *j* | Dragons-blood red to brick red | 1,100.4 | ±20. | 0.000† | *E* | 7.77 | 2 90 |
| 247 | 1,691 | Ferric oxide, dried at 240° | Hydrolysis solution Kahlbaum nitrate, 260° | 5 *k* | Morocco red | 651.9 | ±19. | ...... | ...... | 7 07 | 2 64 |
| 1,367 | 1,637 | Ferric oxide, heated | Merck, heated in vac. furnace, 1,100° | 54′′′′′ *k* | Blackish plumbeous to dark violet-gray | 963. | 22. | 0.094<br>0 1 | *E*<br>*P* | 8.34 | 3.11 |
| 1,072 | 1,692 | Hematite crystals | Deacon process apparatus | 67‴ *m* | Raisin black to taupe brown | 905.9 | ±20. | 0 21 | *P* | 17.47 | 6 51 |
| 1,521 | 1,643 | Oxidized magnetite | Vacuum furnace, 1,200° | 65‴ *l* | Dark slate-purple to raisin black | 442.7 | 23.2 | 0 275<br>0 15 | *E*<br>*P* | 22.2 | 8.28 |
| 1,365 | 1,638 | Merck's ferric oxide, heated | Vacuum furnace, 1,100° | 49′′′′′ *k* | Blackish plumbeous | 434. | 21 6 | 0 23 | *P* | 21.5 | 8 02 |

TABLE 5.—*Ferrous Iron Content and Comparative Susceptibility of Artificial Oxides of Iron in Powder Form.*—(*Continued*)

| Specimen No. | Magnetic Test No. | Designation of Specimen | Source | Color of Powder | | Wt of Sample Tested, Mg. | Temp. | Per Cent. FeO | Method* | Magnetic Attraction per Gram at 800 Milliamp. | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | No. | Name | | | | | In Mg. | Relative |
| 352b | 1,673 | Ferric oxide, ignited over blast lamp | Ferric nitrate, No. 1, from electrolytic iron | 69'''' j | Vinaceous-slate to deep slaty brown | 1,114.2 | ±19. | 0.097 | E | 34.02 | 12.7 |
| 214 | 1,672 | Same, heated in oxygen and air at 1,100°–1,200° | .................... | 49''''' k | Blackish plumbeous | 786.8 | 19.1 | 0.109<br>0.1 | E<br>P | 48.50 | 18.10 |
| 215 | 1,680 | Same, heated in oxygen at 1,100° | .................... | 61''' k | Dark plumbago slate | 1,079.1 | 20. | 0.061 | E | 41.58 | 15.52 |
| 229 | 1,678 | Ferric oxide, ignited over blast lamp | Do. | 69''' l | Anthracene purple to taupe brown | 1,085.1 | ±21. | 0.035 | E | 42.45 | 15.84 |
| 1,523 | 1,644 | Oxidized magnetite | Vacuum furnace, 1,200° | ...... | .................... | 634.4 | 23. | 0.30 | P | 64.0 | 23.9 |
| 1,362 | 1,639 | Merck's ferric oxide, heated | Vacuum furnace, 1,100° | ...... | .................... | 68.3 | 21.6 | 0.37 | P | 76.0 | 28.4 |
| 1,347 | 1,640 | Merck's ferric oxide, heated | Do | ...... | .................... | 150.8 | 22. | 0.55 | P | 128.3 | 47.9 |
| 1,376 | 1,641 | Merck's ferric oxide, heated | Do. | ...... | .................... | 82.9 | 21.5 | 0.77 | P | 136.8 | 51.0 |
| 1,398 | 1,641 | Merck's ferric oxide, heated | Vacuum furnace, 1,200° | ...... | .................... | 63.5 | 22. | 1.17 | P | 256. | 95.5 |
| 1,340 | 1,642 | Merck's ferric oxide, heated | Do., 1,100° | ...... | .................... | 106.7 | ±23. | 1.96 | P | 413. | 153.2 |
| 242 | 1,683 | Oxidized magnetite | Mineville magnetite, quartz, and moist oxygen, 470° | 1'' k | Mineral red | 1,330.4 | ±22. | 2.36 | E | 484. | 180.6 |
| 1,031 | 1,810 | Hematite scales | Clinker from house-heating furnace | 69'''' k | Deep slaty brown | 301.1 | ±18. | 6.48 | P | 818. | 305.1 |
| 1,539 | 1,834 | Oxidized magnetite | Vacuum furnace, 1,200° | 61'''' m | Dull violet-black | 501.5 | ±19. | 4.65 | P | 1,125. | 420. |
| 1,598 | 1,687 | Ferric oxide from nitrate | Do. | 49''''' k | Blackish plumbeous | 932.6 | ±20. | 4.77 | P | 1,185. | 442. |
| | | | | | | 625.9 | | | | 1,179. | 440. |
| 240 | 1,682 | Oxidized magnetite | Mineville magnetite and moist oxygen, 460° | 69'''' k | Deep slaty brown | 969.3 | ±22. | 8.27 | P | 1,878. | 700. |
| 224 | 1,675 | Oxidized magnetite | Bur. Stds. magnetite ore No. 29 and moist oxygen, 400° | 5''' l | Dark vinaceous-brown to seal brown | 1,067.2 | ...... | 12.85 | P | 2,948. | 1,100. |
| 1,548 | 1,832 | Oxidized magnetite | Vacuum furnace, 1,200° | 67''''' k | Dark purplish gray | 488.6 | 17. | 16.11 | P | 4,856. | 1,812. |
| 1,551 | 1,833 | Oxidized magnetite | Do. | 67''''' l | Dark to dusky purplish gray | 479.2 | 18. | 17.57 | P | 5,035. | 1,878. |
| 1,043 | 1,657 | Black oxide | Action of air on pasty iron amalgam | ...... | Black, mat | 9.65 | 21. | 66.1 | P | 2,982. | 1,113. |

* *E*, electrometric titration. *P*, Permanganate titration.
† Contains basic nitrate. Ignition loss 2.41 per cent.
‡ Not homogeneous; portions are as high as 0.043 per cent.

NOTES ON TABLE 5

Samples Nos. 1,041 and 4,478*a* have been described and partial analyses given in *Journal of the American Chemical Society* (1916), **38**, 811.

Nos. 228, 226, 234, 230, and 229 were heated in open platinum crucibles, over the air-gas blast flame, which gives a temperature in such a crucible of about 1,100°. No. 352*b* was similarly heated in an Aupperle crucible.

Nos. 268, 274, 280, 214, and 215 were heated in platinum boats in a fused silica tube inside of a platinum-wound furnace. The apparatus and experiments are described in *Journal of the American Chemical Society* (1916), **38**, 1190.

Nos. 242, 240, and 224 were heated in platinum boats in the same tube furnace as the preceding; No. 242 for 45 hr., No. 240 for 20 hr. and No 224 for 22 hr.

Nos. 231 and 247 were formed by heating a solution of the nitrate for several hours, the former in a sealed Jena combustion tube, the latter in a sealed silica tube, in a bomb furnace. We are indebted to Mr. E. G. Zies of this Laboratory for these heatings. The increase of temperature caused hydrolysis of the nitrate, precipitating the oxide, which settled to the bottom of the tube and was filtered off, washed, and dried.

Nos. 1,367, 1,521, 1,365, 1,523, 1,362, 1,347, 1,376, 1,398, 1,340, 1,539, 1,598, 1,548, and 1,551 were samples which had been used for the determination of dissociation pressures in the series $Fe_2O_3$-$Fe_3O_4$, and are referred to in *Journal of the American Chemical Society* (1916), **38**, 824–826. All had been heated in platinum crucibles in the platinrhodium-tube vacuum furnace.

No. 1,072 was furnished to us through the kindness of Prof. C. E. Munroe, of George Washington University. The sample consists of well-developed platy specular hematite crystals, up to 1 cm. in diameter, which were taken from the pipes of a Deacon-process chlorine apparatus. These crystals and their mode of occurrence have been described by Prof. Munroe in *American Journal of Science* (1907), **24**, 485–486.

No. 1,031 consisted of glistening scales of hematite covering the surface of clinker from a house-heating furnace burning anthracite coal. Some unusual combination of circumstances, perhaps the association of a chloride with an ash high in iron and a moisture-laden air current, caused the deposition of hematite scales within the ash bed. Under the microscope the sample consisted of red scales. A dark residue, amounting to 11.5 per cent. of the original weight, remained after solution in HCl.

No. 1,043 was made by blowing air through a pasty iron amalgam, made by electrolyzing a ferrous sulphate solution with a mercury cathode. The product was shaken up with water, filtered, and washed with alcohol and ether.

were therefore made of a similar mixture which was ground for several hours in an automatic ore-grinding machine with an agate mortar, with the following results:

| | |
|---|---|
| Mineville magnetite No. 646, original........... | 30 84 per cent. FeO |
| Ground 2 hr. 20 min.......................... | 30.14 per cent. FeO |

The finely ground magnetite from the above test was mixed with three times its weight of quartz (between 100 and 200 mesh) and again ground in the automatic grinder:

| | |
|---|---|
| Original mixture.............. ... ......... | 7.53 per cent. FeO |
| Ground 5 hr[9]............................... | 6.76 per cent. FeO |

The mortar was quite warm at the end of the test.

The results show that long-continued grinding causes oxidation of a part of the magnetite. These experiments confirm those of Mauzelius[10] and of Hillebrand[11] concerning the effect of such treatment on the ferrous-iron content of silicate rocks. They found, however, that 15 to 30 min. grinding is usually without serious effect. Our samples were all crushed as far as possible without grinding, and were then ground by hand for 5 to 15 min., with sifting and regrinding of residues. We believe, therefore, that none of the data of Tables 5 and 6 are affected by oxidation due to grinding.

## Experimental Results on Artificial and Natural Oxides

In Tables 5 and 6 we have assembled our data on the comparative susceptibility and the ferrous-iron content of artificial and natural oxides. Table 5 also includes tests on ferrous ammonium sulphate and ferrous sulphate, as a basis of comparison.

The first column of each table gives the reference number of the specimen. For the artificial oxides heated in the vacuum furnace these numbers correspond to those in our previous publications on the oxides of iron. The second column gives the reference number of the magnetic test. The third and fourth columns give the designation and description of the specimen and its source; further notes are given at the ends of the tables. The fifth column gives the color of the powdered substance when pressed flat in the glass testing-capsule. The color names and numbers are according to Ridgway.[12] In the sixth column is the weight of oxide

[9] The product with 6.76 per cent. FeO served as the original material for sample No. 242 of Table 5.

[10] R. Mauzelius: The Determination of Ferrous Iron in Rock Analysis. *Sveriges Geologiska Undersokning*, Arsbok (1907), No. 3.

[11] W. F. Hillebrand: The Influence of Fine Grinding on the Water and Ferrous-iron Content of Minerals and Rocks. *Journal of the American Chemical Society* (1908), **30**, 1120–1131. The Analysis of Silicate and Carbonate Rocks. *U. S. Geological Survey, Bulletin* 422 (1910), 53.

[12] *Color Standards and Color Nomenclature*, Washington, 1912.

tested magnetically, and in the seventh the temperature. No correction was made for temperature variation, as the experimental error is in most cases as great as the correction to an average temperature. The eighth column contains the percentage of FeO on the total weight of sample, as determined analytically, followed (in Table 5 only) by a column indicating the method of analysis used. *E* indicates electrometric titration, *P* indicates titration with potassium permanganate. The last two columns give the magnetic attraction per gram of sample under the standard conditions, in milligrams as well as in multiples of the attraction on ferrous ammonium sulphate.[13]

### *Discussion of Results on Artificial Oxides*

The data of Table 5 are shown graphically in Fig. 3. In order not to crowd the samples low in FeO into an illegible jumble in the corner of the diagram, we have plotted the logarithm of the magnetic pull against the logarithm of the ferrous iron percentage. The curve of the quartz-magnetite mixtures of Table 4 is also included in Fig. 3.

The individual samples of Table 5 are arranged in the order of increasing magnetic susceptibility. From the table and from the curve of Fig. 3 it is evident that most of the artificial solid-solution samples have a magnetic attraction approximately proportional to their ferrous iron content. The possibility, suggested by the action of a hand magnet on the powders, that the attraction might increase much more rapidly than in proportion to the ferrous iron percentage, was not realized. From the standpoint of their magnetic susceptibility, these oxides are therefore indistinguishable from mixtures of magnetite and paramagnetic hematite. It is possible, however, that other magnetic properties will show a distinction between the two modes of occurrence of the ferrous iron.

Where the ferrous iron is present in some other form than in solid solution or in admixed magnetite, the corresponding magnetic attraction will fall below the mixture line. No. 1,031, for instance, which lies well below the line, undoubtedly contains considerable ferrous silicate.

When the percentage of FeO goes below about 0.2 per cent. the results are much less regular. There is a general drift of proportionality between FeO content and susceptibility, but other factors evidently become of importance. The variation in susceptibility is far too large to be accounted for by the FeO content or by any other of the known factors. The two samples made by hydrolysis of ferric nitrate form an

[13] Ferrous ammonium sulphate was adopted as the standard substance, not only because it is the purest and most easily reproducible salt of ferrous iron, but also because its magnetic susceptibility in the crystalline form has been carefully determined for all three crystallographic directions (W. Finke: *Annalen der Physik* (1910), **31,** 149–168).

extreme instance of this, as they contain no determinable ferrous iron, yet agree in having a susceptibility which is four times as high as that of the least magnetic oxide which contains determinable FeO. It is still possible, however, in this case, that some FeO existed in the presence of the excess of nitrate, and that it was oxidized during solution in HCl.

The four samples numbered 352*b*, 214, 215 and 229 are abnormally high in susceptibility for some as yet undiscovered reason. The cause evidently lies in the original ferric nitrate from which the samples were made. This nitrate was evaporated directly to dryness after solution of the electrolytic iron in nitric acid, and was not filtered or crystallized; a minute impurity in suspension, consisting of some undissolved and unoxidized substance like ferrosilicon, might explain the abnormal results. Oxides made from filtered and crystallized nitrate from the same iron are normal.

There is one striking exception to the general proportionality of magnetic susceptibility and ferrous-iron content, namely, the so-called "magnetic ferric oxide."[14] This can be easily prepared by the oxidation of precipitated magnetite at atmospheric temperatures, and is practically as ferromagnetic as magnetite itself. It loses its high susceptibility in a few minutes at 750°, and does not regain it on cooling. Slow oxidation of natural magnetite at 400° to 500° yields a non-magnetic oxide, as is evident from Nos. 242, 240 and 224, all of which have a susceptibility in normal agreement with their ferrous-iron content. Further experiments on the magnetic form of ferric oxide are in progress.

The oxide No. 1,043, which lies between $Fe_3O_4$ and FeO in composition, is considerably less magnetic than $Fe_3O_4$. This is in qualitative agreement with the observations of Hilpert and Beyer.[15]

### *Colors of the Artificial Oxides*

The color of the powdered oxide was observed in most cases at the same time as the magnetic susceptibility. The color is stated in terms of Ridgway's standards.[16] A few words will serve to indicate the basis of the color system used by Ridgway. The pure spectrum colors and intermediate hues (including a series between violet and red) are numbered from 1 to 72. Tints, marked *a* to *g*, are made from these by admixture of white, and shades, marked *h* to *n*, by admixture of black. Each of these is further dulled by admixture of neutral gray, to form five series, indicated by accent marks from ′ to ′′′′′.

[14] S. Hilpert: Über Beziehungen zwischen chemischer Konstitution und magnetischen Eigenschaften bei Eisenverbindungen. *Berichte Deutschen Physikalische Gesellschaft* (1909), **11**, 293–299.

[15] S. Hilpert and J. Beyer: Über Eisenoxyduloxyde und Eisenoxydul. *Berichte Deutschen Chemischen Gesellschaft* (1911), **44**, 1608–1619.

[16] Robert Ridgway: *Color Standards and Color Nomenclature*, Washington, 1912.

Practically all the iron oxide hues in Table 5 lie in the range 65 to 72 and 1 to 5. No. 65 is a reddish red-violet and No. 5 is an orange orange-red. The oxide colors include no tints (tones lighter than the pure hues), and the shades (tones darker than pure hues) lie between *j* and *m* (59 to 87.5 per cent. black). There is only one example of a pure bright color, and only one example of the first degree of dullness (32 per cent. gray); the remainder are dulled by 58 to 95 per cent. gray.

The purer ferric oxides approach the pure red and orange-red colors, while those high in ferrous iron approach the darker shades of the same colors. The fineness of the powder has a much greater effect upon the color than the ferrous iron percentage. Samples 231 and 247, which are agglomerated colloids and extremely finely divided, furnish the two examples of the brighter colors referred to in the preceding paragraph. Equally pure oxides which have been sintered by heating at 1100° or 1200° go over toward the violet, although fine grinding brings their color back toward red.

These facts are in agreement with the microscopic characters of the oxides. Pure $Fe_2O_3$ in thin flakes is deep red in color, and the color is darkened by solution of $Fe_3O_4$. The absorption is large, so that with increasing thickness the particles soon become opaque; the result is a change in color of the powder toward the violet.

### *Discussion of Results on Natural Oxides*

The data of Table 6 are plotted in Fig. 4. As in Fig. 3, the scales of both magnetic attraction and percentage of FeO are logarithmic, to avoid the crowding of part of the data into one corner of the diagram. The quartz-magnetite curve of Fig. 3 is reproduced in Fig. 4.

The natural specimens in Table 6 are arranged in the order of increasing percentage of FeO. They were selected to give a wide variety of ferrous oxide percentages and a wide variety of modes of origin, rather than to show the variations in any particular locality or type of occurrence. Nearly all of them are clean and well-crystallized specimens. We hope later to devote a more detailed study to carefully located specimens from particular localities, as it is only in this manner that the data necessary for definite and detailed conclusions can be obtained. But there are certain general relations shown in Table 6 and Fig. 4 which should be pointed out.

1. If the natural solid solutions and mixtures of magnetite and hematite correspond in properties to the artificial, all of the points of Fig. 4 should be found on or below the quartz-magnetite mixture curve. With one exception, this is found to be the case. The single exception is the volcanic sublimate from Madeira; the cause of its high magnetism has not yet been found.

TABLE 6.—*Ferrous Iron Content and Comparative Susceptibility of Natural Oxides in Powder Form*

| Specimen No. | Magnetic Test No. | Designation of Specimen | Source | Color of Powder | | Wt of Sample Tested, Mg. | Temp. | Per Cent. FeO | Magnetic Attraction per Gram at 800 Milliamp. | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | No. | Name | | | | In Mg | Relative |
| 1,058 | 1,601 | Specular hematite | Lake Superior | 5 *k* | Morocco red | 1,010.6 | ±19. | 0.15* 0.167† | 14.88 | 5.55 |
| 1,090 | 1,803 | Hematite, twinned | Dognacska, Hungary | 65‴ *k* | Dark slate-purple | 714.8 | ±20. | 0.20 | 20.0 | 7.16 |
| 1,011 | 1,814 | Specular hematite | Puy de Dome, France | 69‴ *l* | Anthracene purple to taupe brown | 792.9 | ±21. | 0.28 | 18.28 | 6.82 |
| 1,026 | 1,808 | Specular hematite | Cumberland, England | 65‴ *l* | Dark slate-purple to raisin black | 1,116.3 | ±22. | 0.31 | 28.0 | 10.15 |
| 1,027 | 1,808 | Turgite portion of above | ........................ | 5′ *k* | Brick red | 964.8 | ±22. | 0.36 | 5.42 | 2.02 |
| 1,076 | 1,803 | Specular hematite | Altenberg, Saxony | 65‴ *l* | Dark slate-purple to raisin black | 1,211.0 | ±20. | 0.33 | 25.2 | 9.40 |
| 1,074 | 1,804 | Specular hematite | Island of Madeira | 65‴ *k* | Dark slate-purple | 1,645.6 | ±20. | 0.39 | 142.1 | 53.1 |
| 1,044 | 1,686 | Limonite | Ktaadn Iron Works, Maine | 15′ *i* | Ochraceous-tawny | 1,053.7 | 22. | 0.62 | 3.06 | 1.11 |
| 1,016 | 1,685 | Micaceous hematite | Pennsylvania | 67‴ *m* | Raisin black to taupe brown | 601.8 | 21.9 | 0.65 | 15.09 | 5.63 |
| 1,028 | 1,684 | Martite | Twin Peaks, Utah | 65‴ *l* | Raisin black to dark slate-purple | 1,152.6 | 22 | 0.79 | 97.8 | 36.5 |
| 1,028 | | Same, fractionated: | | | | | | | | |
| | 1,817 | (*a*) Magnetic fraction | ........................ | 69‴ *l* | Anthracene purple to taupe brown | 515.4 | ±22. | 1.13 | 193.6 | 72.2 |
| | 1,817 | (*b*) Residual fraction | ........................ | 67‴ *m* | Raisin black to taupe brown | 1,029.5 | ±22 | 0.53 | 17.99 | 6.71 |
| 1,052 | 1,816 | Martite, dodecahedral | Nova Scotia | 69‴ *m* | Taupe brown | 988.0 | ±21. | 1.44 | 191.2 | 71.3 |
| 1,037 | 1,812 | Hematite, crystalline | Dickerson, Md. | 67‴ *m* | Raisin black to taupe brown | 437.6 | ±19 | 1.46 | 13.62 | 5.08 |
| 1,086 | 1,806 | Specular hematite | Guanajuato, Mex. | 69‴ *l* | Anthracene purple to taupe brown | 381.1 | 22 | 1.73 | 38.5 | 14.37 |
| 1,092 | | Iron ore: | Juragua, Cuba | | | | | | | |
| | 1,804 | (*a*) Most magnetic fraction (10 per cent.) | ........................ | 69‴ *k* | Anthracene purple | 303.1 | ±21 | 2.59 | 495 | 184.7 |
| | 1,805 | (*b*) Less magnetic fraction (46 per cent.) | ........................ | 69⁗ *l* | Anthracene purple to taupe brown | 1,038.9 | ±21 | 0.67 | 58.2 | 21.7 |
| | 1,805 | (*c*) Least magnetic fraction (44 per cent.) | ........................ | 71″ *m* | Mars violet | 1,052.1 | ±21 | 0.37 | 10.85 | 4.05 |
| 1,047 | 1,820 | Martite | Cleveland Mine, Michigan | 69‴ *l* | Anthracene purple to taupe brown | 538.0 | ±21. | 3.13 | 582. | 217.1 |
| 1,022 | 1,819 | Martite | Twin Peaks, Utah | 69⁗ *l* | Deep slaty brown to aniline black | 535.0 | ±21. | 10.53 | 2,078 | 775 |
| 1,051 | 1,815 | Martite | Iron Hill, Colo. | 69‴ *m* | Taupe brown | 982.1 | ±22 | 10.37 | 2,620. | 977 |
| 1,094 | 1,821 | Iron ore | Daiquiri, Cuba | 69⁗ *l* | Deep slaty brown to aniline black | 553.3 | ±20 | 12.96 | 2,739 | 1,022 |

(*Continued on following page.*)

TABLE 6.—*Ferrous Iron Content and Comparative Susceptibility of Natural Oxides in Powder Form.*—(*Continued*)

| Specimen No. | Magnetic Test No. | Designation of Specimen | Source | Color of Powder | | Wt. of Sample Tested, Mg. | Temp. | Per Cent. FeO | Magnetic Attraction per Gram at 800 Milliamp. | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | No. | Name | | | | In Mg. | Relative |
| 1,012 | | Martite: | Minas Geraes, Brazil | | | | | | | |
| | 1,811 | (*a*) Magnetic fraction (8 per cent.) | ...................... | 67‴ *m* | Raisin black to taupe brown | 217.1 | ±20. | 15.33 | 3,850. | 1,437. |
| | 1,811 | (*b*) Residual fraction (92 per cent.) | ...................... | 71″ *m* | Mars violet | 1,135.9 | ±20. | 0.65 | 34.63 | 12 92 |
| 1,084 | 1,807 | Hematite | Tavetsch, Switzerland | 57⁗ *m* | Blue-violet black | 418.7 | 22. | 15.8 | 3,936. | 1,468 |
| 1,050 | 1,818 | Martite | Woody, Cal. | 69⁗ *m* | Aniline black | 509.1 | ±21. | 17.75 | 4,500. | 1,678 |
| 1,835 | 1,835 | Ilmenite | Arendal, Norway | 65′′′′′ *m* | Dusky purplish gray | 500.0 | .... | ..... | 84.8 | 31 6 |
| 3*b* | 1,828 | Magnetite, 0.05 mm. grains and finer | Mineville, N. Y. | ....... | Black, lustrous | 500.1 | ±19. | 29.62† | 8,718. | 3,253. |
| 3*b* | 1,825 | Magnetite, 0.16 mm. grains, average | Do. | ....... | Black, lustrous | 500.1 | ±20. | 29.66† | 9,516. | 3,551. |

* Permanganate titration. All of the analyses in this table were made by this method except those marked †.
† Electrometric titration.

NOTES ON TABLE 6

1,011. Specular hematite, Puy de Dome. Foote Mineral Co., No. 825. A light brown trachytic tuff, impregnated with glistening hematite crystals up to 1 mm. in diameter. A portion was crushed and cleaned magnetically, and the crystals were washed with water. A dark gray residue, amounting to 6.3 per cent. of the original weight, remained after solution of the sample in HCl for analysis.

1,012. Martite, Minas Geraes, Brazil. Foote Mineral Co., No. 843. Loose, clean, bright octahedra, 1 to 4 mm. in diameter. Contains minute inclusions, probably rutile. Microscopic structure shows that the octahedra are pseudomorphs.

1,016. Micaceous hematite, unknown locality in Pennsylvania. Foote Mineral Co., No. 833. Radiating curving bundles of very thin leaves, associated with quartz. Magnetically homogeneous. Very slight residue, containing quartz, left after solution in HCl.

1,022. Martite, Twin Peaks, Millard Co., Utah. Ward's Natural Science Establishment. Intergrown octahedra, up to 25 mm. on edge. No evidence of any hematite-magnetite intergrowth visible under microscope. Powder is magnetically homogeneous, except for a slight residue containing apatite. Slight residue of silica left after solution in HCl.

1,026–1,027. Hematite, Cumberland, England, Foote Mineral Co., No. 821. Tabular crystals, associated with clear well-developed quartz crystals, on a red botryoidal base. The botryoidal portion has the microscopic characteristics of turgite. Powder contains several per cent. of calcite, whose high index indicates $FeCO_3$ in solid solution.

1,028. Martite, Twin Peaks, Millard Co., Utah. Foote Mineral Co., No. 845. An altered quartzitic rock fragment covered on both sides with bright octahedra, 1 to 4 mm. in diameter. The octahedra are smaller on one side of the fragment than on the other. The larger octahedra contain some crystals of pyroxene. The larger crystals were powdered, and fractionated magnetically.

1,037. Hematite, Dickerson, Md. R. B. Sosman, No. 203. Intergrown with epidote and garnet. Occurs in Triassic calcareous shale, about 5 meters from west contact of a vertical diabase dike, about 100 meters wide. A slight residue remained after solution in HCl.

1,044. Limonite, Ktaadn Iron Works, Maine. G. P. Merrill, U. S. National Museum, No. 65,331. Occurs in loose porous form, frequently replacing the organic matter of logs, twigs, and leaves and forming "pseudomorphs" of decaying vegetation. The source of the iron is the pyrrhotite of nearby rocks, which weathers, giving rise to iron-bearing waters which flow into a bog and there deposit the limonite.

1,047. Martite, Cleveland Mine, Michigan. Foote Mineral Co., No. 232*r*. Bright octahedra, up to 1 mm. in diameter, thickly disseminated through fine-grained dark greenish-gray chlorite schist. Octahedra were separated from crushed rock by hand magnet.

1,050. Martite, near Woody, Kern Co., Cal. E. S. Larsen, No. W4. Loose-textured dark brown iron ore. A contact-metamorphic product, associated with granite. Sample is magnetically homogeneous, except for a slight residue of yellowish-brown limonite.

1,051. Martite, Iron Hill, Uncompahgre quadrangle, Gunnison Co., Colo. E. S. Larsen, No. U251. Octahedra, up to 15 mm. on edge, in loose brown matrix. Occurs on the Beaver Creek side of Iron Hill, in a martite-apatite dike about 7 meters wide, cutting limestone.

1,052. Martite, Nova Scotia. U. S. National Museum, No. 81,386. Dodecahedra, up to 4 mm. in diameter, in a reddish matrix, associated with crystalline quartz. The dodecahedral form is unusual.

1,058. Specular hematite, Lake Superior region. Foote Mineral Co., No. 232.

Occurs as velvety-appearing botryoidal lining in cavities in massive ore. A small amount of dark siliceous residue remained after solution in HCl.

1,074. Hematite, Island of Madeira. Collected by J. D. Dana U. S. National Museum, No. 13,388. A fragment of reddish-brown basic igneous rock, encrusted with hematite scales up to 1 mm. in diameter A volcanic sublimate. Incomplete solution in HCl leaves irregular hematite grains with projecting spikes. Solution in HCl leaves a residue of 2.6 per cent. of a silicate. containing inclusions.

1,076. Specular hematite, Altenberg, Saxony. Foote Mineral Co. Quartzitic rock containing stringers of black hematite, with free-growing crystals up to 2 mm. in diameter.

1,084. Specular hematite, Tavetsch, Switzerland. Foote Mineral Co. Hematite plates up to 2 mm. in diameter, on a joint face of a fragment of mica schist. Among the hematite crystals are some microscopic octahedra with rusty surfaces. About 3 per cent. siliceous residue remains after solution in HCl.

1,086. Specular hematite, Guanajuato, Mex. Foote Mineral Co. A fragment of rhyolite encrusted with hematite crystals up to 1 mm. in diameter. A bright-red residue, uniaxial and of high refractive index, probably a form of $TiO_2$, remains after solution in HCl.

1,090. Hematite, twinned, Dognacska, Hungary. Foote Mineral Co., No. 829. Crystals up to 3 mm. in diameter, associated with well-developed pyrite cubes, on massive magnetic ore. Crystals are twinned according to $10\bar{1}0$.

1,092. Iron ore, Juragua, Cuba. W. Lindgren, No. 12 Juragua. From stock pile at Santiago Bay. Uniform blue-black glistening hematite ore, with few red partings. The ore is described by Lindgren and Ross in *Trans.* (1916), **53**, 55–58

1,094. Iron ore, Daiquiri, Cuba. W. Lindgren, No. 10 Daiquiri. From San Antonio lowest open cut. Dull gray ore with light yellow spots and partings. The ore is described in the article cited under 1,092, pp. 43–52.

2. If the FeO in a natural sample is not in solid solution in $Fe_2O_3$, or is not in the form of admixed magnetite, then its corresponding point in Fig. 4 should fall below the mixture line. The specimen which falls relatively farthest below this line is No. 1,044, the limonite from Maine, in which practically all of the ferrous iron is in some form not in solid solution in the $Fe_2O_3$, probably as $FeCO_3$; this sample formed at atmospheric temperatures. No. 1,037, which formed in shale near a diabase dike, and No. 1,027 (the turgite portion of No. 1,026) come next in order. Both of these have probably been formed at lower temperatures than the highly ferrous specular hematites. About equally distant below the line is the Guanajuato hematite, No. 1,086, in which some of the FeO evidently loses its magnetism through combination with $TiO_2$. Various martites and specular hematites occur still nearer the mixture line.

3. The oxides higher in FeO lie much nearer the line than do those lower in FeO. The higher percentages of FeO in the former group, coupled with their more nearly normal magnetism, may indicate a higher temperature of formation of these oxides. Further experimental study of their conditions of formation and of their various magnetic properties will be necessary before final conclusions can be drawn concerning their origin.

4. Several of the specimens were fractionated magnetically (by the method described in the following paper), and the fractions were tested

and analyzed separately. In each case the less magnetic portion falls farther below the line than the more magnetic portion.

An interesting case of this kind is the Juragua iron ore, No. 1,092. The points representing the three fractions of the Juragua ore are connected by a broken line in Fig. 4. It is difficult to reconcile this graded

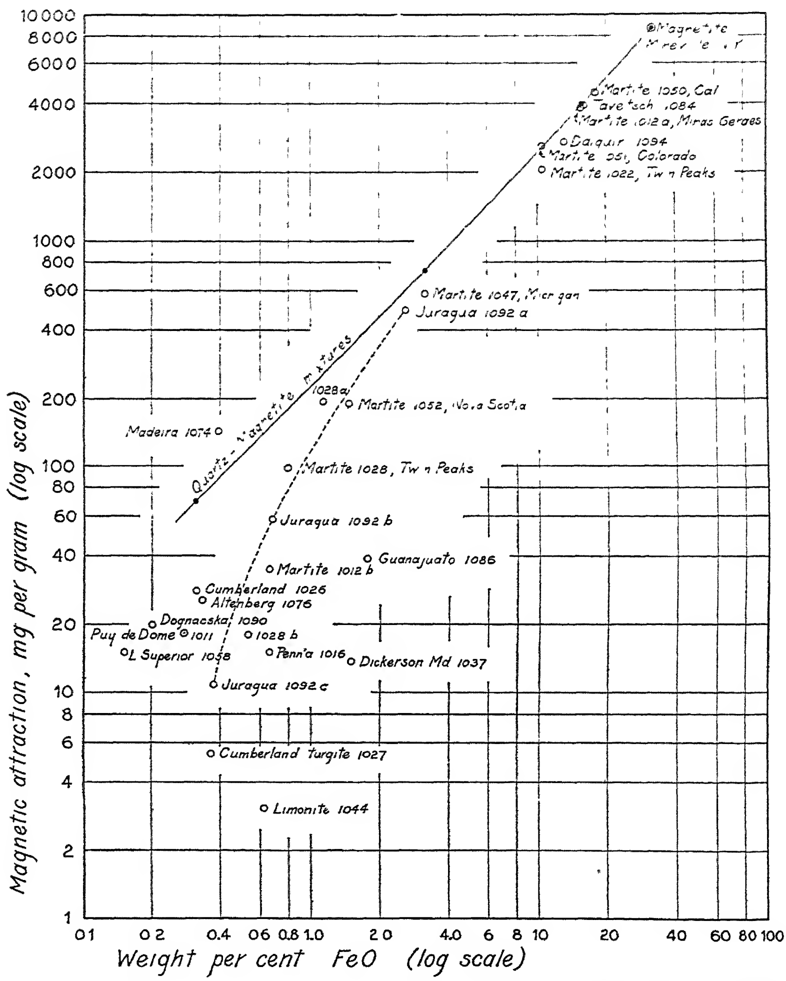


FIG. 4.—RELATION OF MAGNETIC ATTRACTION TO PERCENTAGE OF FeO, NATURAL OXIDES.

deviation of the successive fractions away from the mixture line with Lindgren and Ross's theory[17] of the origin of the South Cuban ores, according to which the ore was originally magnetite which has been oxidized by oxygen from the surface. If the ore were simply a mixture of

[17] W. Lindgren and C. P. Ross: *Trans.* (1916), **53**, 40–59.

hematite and unoxidized magnetite, all of the fractions should lie on the mixture line or deviate from it by a constant amount. But if the ore had been deposited by a vapor or solution of continually changing properties, so that the resulting product changed gradually in composition as the deposition (or alteration of previously deposited ore) proceeded, we might reasonably expect just the kind of gradation in magnetic properties that is actually found in the Juragua ore. This question will be considered further in the following paper on Zonal Growth in Hematite.

It should be noted that although both the Daiquiri and the Juragua ore specimens are "magnetic," in the usual sense, they are very far from being magnetites. Even the most magnetic fraction of the Juragua specimen contains only 2.59 per cent. FeO, whereas magnetite contains 31.03 per cent.

### *Colors of the Natural Oxides*

The colors of the powdered natural oxides cover approximately the same range as the colors of the artificial oxides (see Tables 5 and 6). Only two natural specimens, the Lake Superior hematite (1,058) and the Cumberland turgite (1,027) possess the pure colors characteristic of the hydrolyzed products in Table 5 (Nos. 231 and 247). Most of the other natural oxides are of the violet-red shades (65 to 71) characteristic of the artificial oxides which have been crystallized or sintered. Limonite 1,044, a hydrated oxide, is entirely different in color from the anhydrous oxides, being of a yellow-orange shade.

TABLE 7.—*Colors of Fractions of Elba Hematite Crystal*

| Portion | Percentage of FeO | Color No. | Color Name |
|---|---|---|---|
| *a* | 0.29 | 63‴ *m* | Dull violet-black to raisin black |
| *b* | ..... | 65‴ *l* | Raisin black to dark slate-purple |
| *c* | ..... | 67‴ *m* | Raisin black to taupe brown |
| *d* | 0.58 | 67‴ *k* | Dark slate-purple to anthracene purple |
| *e* | ..... | 69‴ *l* | Anthracene purple to taupe brown |
| *f* | 4.24 | 67‴ *m* | Raisin black to taupe brown |
| *g* | ..... | 65‴ *l* | Raisin black to dark slate-purple |
| *h* | ..... | 69‴ *l* | Anthracene purple to taupe brown |
| *i* | 6.76 | 67‴ *m* | Raisin black to taupe brown |
| *j* | ..... | 65‴ *m* | Raisin black |
| *k* | 3.31 | 63‴ *m* | Dull violet-black to raisin black |
| *l* | ..... | 65‴ *m* | Raisin black |
| *m* | ..... | 67‴ *m* | Raisin black to taupe brown |

The fact that fineness of grain, rather than chemical composition, has the predominating influence upon the color, is shown in Table 7, in which are given the colors of the powders of the various fractions of an Elba hematite crystal (which will be further described in the paper which follows). The specimens were all crushed to pass a 100-mesh sieve (open-

ing 0.140 mm.). Although the ferrous iron in these fractions covers the wide range from about 0.2 to 6.76 per cent. FeO, nevertheless the hues all lie within the range 63 to 69, the shades are all within the range *k* to *m*, and the dullness of all the samples is the same (expressed by ‴, or 77 per cent. gray). These colors were measured on different days, and without comparison of one powder with another, but only by direct comparison with the color standards.

## Martite

The mineral martite is generally considered to be a pseudomorph of hematite after magnetite. It frequently has the composition of nearly pure $Fe_2O_3$, while retaining the isometric (usually octahedral) external form of magnetite. But many specimens, like several of those in Table 6, for example, contain large percentages of FeO, and are magnetically homogeneous, suggesting the possibility that they may be true solid solutions of $Fe_3O_4$ in $Fe_2O_3$.

These martites are found in fact to be both pseudomorph and solid solution at the same time. Their fibrous or granular character shows that they are real pseudomorphs, but the fibers or granules themselves appear to consist of a solid solution of $Fe_3O_4$ in $Fe_2O_3$.

The following descriptions[18] of the microscopic characters of three natural martites whose ferrous iron content and magnetic susceptibility have been given in Table 6 will illustrate the properties of this mineral.

No. 1,012. Martite from Minas Geraes, Brazil. The octahedra are fine-grained aggregates of hematite-colored, strongly doubly refracting granules, and are therefore pseudomorphic.

No. 1,047. Martite from Cleveland Mine, Michigan. Aggregates similar to those of No. 1,012.

No. 1,022. Martite from Twin Peaks, Utah. In powder the grains appear to be single units optically, but thin sections show that these grains are fragments of larger platy crystals which form roughly radiating aggregates within the octahedral crystals. These aggregates occupy the surfaces as well as the interiors of the octahedra. This, also, is a case of pseudomorphism. The grains in the powder are found to be uniaxial and negative. $\epsilon$ for wave-length $700\mu\mu$ was observed to be 2.635. The double refraction for this wave-length (as determined by observing the maximum number of orders of interference colors around grains about 0.01 mm. thick) is about 0.230.

Although this martite is optically somewhat similar to hematite, the refractive indices are much lower. The lowering can be due only partly to FeO, if at all, for the martite is about as light-colored as pure hematite. The refractive indices are intermediate between hematite and the fibrous water-containing form of $Fe_2O_3$ known as turgite, in which the usual

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[18] Microscopic Examinations by H. E. Merwin.

values of the refractive indices are: $\epsilon_{Li} = 2.37$ to 2.47 amd $\omega_{Li} = 2.48$ to 2.57.

Pleochroism is not noticeable, although an apparent stronger absorption of $\omega$, caused chiefly or entirely by reflection from the irregular surfaces of grains immersed in a substance having the refractive index $\epsilon$, was observed.

*Origin of Martite*

The solid solutions that would be formed at temperatures approaching atmospheric, and by water solutions that were oxidizing rather than reducing in character, would be high in ferric iron and low in ferrous. The alteration of magnetite at a relatively low temperature and under oxidizing conditions should therefore yield a martite high in ferric iron. On the other hand, formation from magnetite at a relatively high temperature is strongly suggested by the properties of those martites that are high in ferrous iron, and which are at the same time homogeneous.

There is another possible criterion which can be applied to martite. The oxidation of artificial magnetite at ordinary temperatures in the laboratory yields a ferric oxide which is practically as ferromagnetic as magnetite itself (see page 423). This high magnetic susceptibility is lost in a few minutes at 750°, but there is no evidence as yet to show that it disappears or even diminishes at atmospheric temperatures. On the contrary, we have found one undoubted natural example of it which must have persisted for a period of time measurable in geologic terms. It appears probable,[19] also, that the oxidation of natural crystalline magnetite at low temperatures yields a magnetic $Fe_2O_3$. If, therefore, the magnetism of a given natural iron oxide is found to be in normal agreement with its content of ferrous iron, the natural inference is that either (1) this percentage of ferrous iron is original, or (2) the oxide has been subjected to a temperature high enough to have brought its magnetic susceptibility down to the normal value corresponding to its ferrous iron content. How high this temperature would need to be under geological conditions remains to be determined.

What was the agent and the process of the undoubted alteration of many magnetites into martites? In considering this question, we are brought face to face with the problem of the permeability of minerals to liquids or vapors. The conversion of magnetite to martite is so complete and uniform in many cases, yielding "crystals" which have no core or other evidence of alteration from the outside inward, that one is strongly

[19] War emergencies have temporarily interrupted the work and have prevented our obtaining complete proof of the oxidation of natural crystalline magnetite to magnetic $Fe_2O_3$. Magnetite is so resistant to oxidizing agents that it is difficult to bring the FeO down to zero at a temperature which will not destroy the magnetism of the $Fe_2O_3$. We have, however, succeeded in oxidizing Mineville magnetite to an oxide, which, although it still contains an appreciable percentage of FeO, is much more magnetic than a normal oxide of the same composition.

tempted to believe that the altering solutions passed straight through the body of the magnetite. We know that metallic iron at a red heat is permeable to carbon monoxide, that hot platinum is permeable to hydrogen, and that hot silver is permeable to oxygen. Is there a temperature at which magnetite is permeable to water vapor or to water solutions of oxidizing agents? This question of permeability need not be confined to magnetite, for petrographic evidence is continually suggesting that solutions or vapors have at some time passed directly through minerals which under atmospheric conditions are impermeable.

## Summary

The percentage of ferrous iron and the relative magnetic susceptibility in powder form have been determined on a number of artificial and natural oxides of iron. Artificial oxides made at 1100° and 1200° consist of a solid solution of $Fe_3O_4$ in $Fe_2O_3$. Their relative magnetic susceptibility is approximately proportional to their percentage of FeO, from $Fe_2O_3$ over to $Fe_3O_4$. The deviations may be partly accounted for by the effect of various factors, of which the fineness of grain of the powdered oxide is the most important, especially in the case of the more ferromagnetic members of the series. The colors of the powdered oxides depend both on their chemical composition and on their physical constitution, especially the fineness of grain.

In addition to the oxides whose susceptibility depends upon their content of FeO, there exists also a highly ferromagnetic form of $Fe_2O_3$, which appears to be rare in natural occurrence.

The natural iron oxide minerals are similar to the artificial in being in many cases solid solutions of $Fe_3O_4$ in $Fe_2O_3$. Others are mixtures of $Fe_3O_4$ and $Fe_2O_3$. If the ferrous iron is not in solid solution or in magnetite admixture, the magnetic susceptibility falls below the normal.

Some natural oxides can be magnetically fractionated; in these cases the less magnetic portions are found to deviate more widely from normal than the more magnetic. The cause of this deviation is not yet entirely clear.

Martite is a pseudomorph after magnetite, but its constituent granules or fibers consist usually of a solid solution of $Fe_3O_4$ in $Fe_2O_3$. The ferrous iron content and the magnetic susceptibility of the specimens examined suggest that they have been produced at temperatures considerably higher than atmospheric.

We wish to acknowledge our indebtedness to Messrs. Merrill, Wherry, Larsen, Lindgren, and Munroe for specimens of oxides of iron; to Mr. C. W. Burrows of the U. S. Bureau of Standards for suggestions concerning the magnetic tests; and to our colleagues of the Geophysical Laboratory, especially to Mr. Merwin for microscopic examinations and to Mr. H. S. Roberts for the electrometric titrations.

# Zonal Growth in Hematite, and Its Bearing on the Origin of Certain Iron Ores

BY R. B. SOSMAN* AND J. C. HOSTETTER,* WASHINGTON, D. C.

(St Louis Meeting, October, 1917)

## Introduction

We have shown in the preceding paper that practically all natural oxides of iron contain a determinable percentage of ferrous iron, and in many cases the percentage approaches that in magnetite itself. Some of these oxides are true homogeneous solid solutions. Others are not homogeneous, and can be separated magnetically into fractions. The nature and cause of this inhomogeneity in certain special cases form the subject of the present paper.

## Magnetic Fractionation of Oxides of Iron

The magnetic fractionation of a sample of oxide can be conveniently carried out by powdering the material very fine, and exposing it at various distances from the poles of an electromagnet. The magnet that we used was specially designed[1] to give a strong field together with a steep gradient outside of the poles. It is set in a frame, with the pole pieces directed downward. A glass plate can be adjusted at various distances beneath the poles. If the fine iron oxide powder is placed just beneath the glass plate, successive portions of the powder are picked up by the magnetic attraction and are caught against the glass plate, the most magnetic portions being taken up at the greatest distance from the poles. Some natural oxides can thus be separated into several fractions of different magnetic susceptibilities.

Among other natural iron oxides, we examined the powder obtained by crushing some crystals of hematite from Elba.[2] We found that

* Geophysical Laboratory, Carnegie Institution of Washington.

[1] This magnet was designed by C. W. Burrows, of the Bureau of Standards, for F. E. Wright of this Laboratory.

[2] Elba hematite crystals are well known in museums for their large size and excellent development. It is said that a great many of those now in American collections were found in shipments of Elba ore which came into the port of Philadelphia up to the year 1895.

the powder contained a notable percentage of ferrous iron, and also that it could be fractionated magnetically in the manner described above. The possibility of fractionation shows that the crystal is not homogeneous, as would be the case if it were a uniform solid solution throughout. It

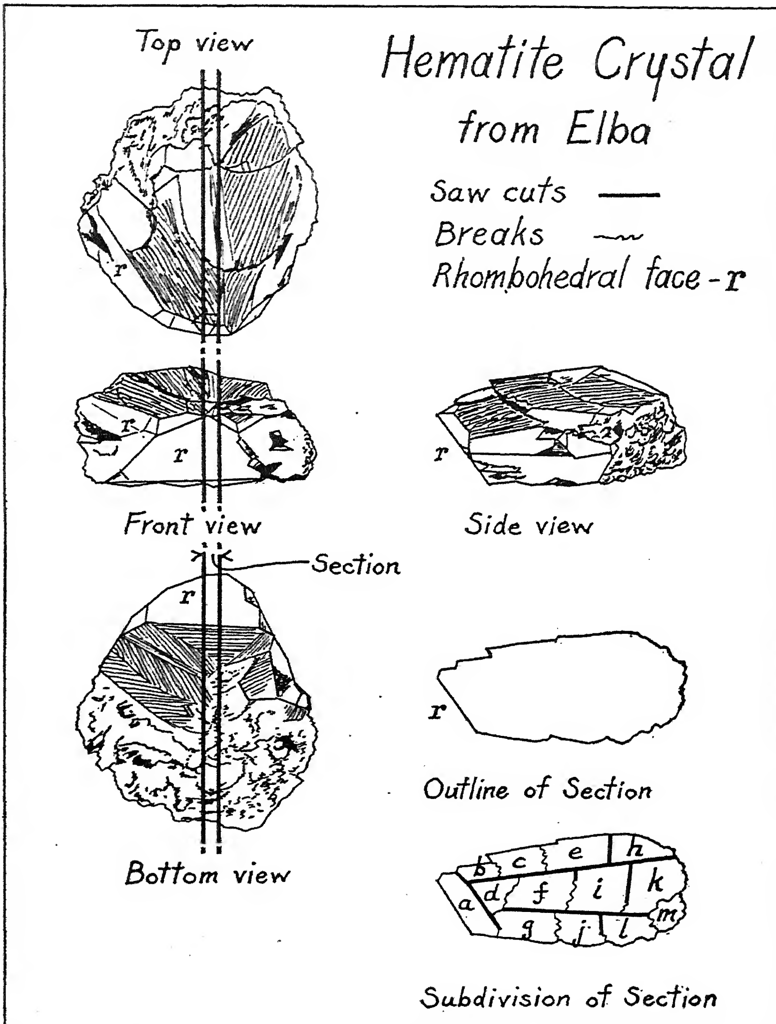


FIG. 1.—SECTIONING OF ELBA HEMATITE CRYSTAL FOR MAGNETIC TESTS.

is immediately evident that there may be two possible explanations of this inhomogeneity:

1. The crystal may be uniform in gross composition, but at the same time heterogeneous, to the extent that it consists of an intimate mixture

of very small crystals of hematite and magnetite, or of two or more other intermediate oxides differing in their percentage of FeO and in their magnetic susceptibility; or

2. The percentage of FeO and the magnetic susceptibility may vary in some continuous manner along the axes of the crystal.

We sought to decide between these alternatives by cutting a thin slice through the middle of a large Elba crystal, and then cutting this slice up into small fragments. Each of these was tested separately, and the results correlated with the position of the fragment in the crystal.

The shape of the crystal and the location of the sections are shown in Fig. 1, drawn approximately to scale, and oriented by a comparison of the faces with crystallographic descriptions of the crystals in Dana's *System of Mineralogy.*

### *Apparatus and Method*

Each of the lettered fragments of Fig. 1 was powdered to pass a 100-mesh sieve (opening 0.140 mm.), and the powder was tested magnetically by measurement of the pull in milligrams exerted on a known weight of it by the field of an electromagnet, at a definite value of the magnetizing current. Absolute measurements of the magnetic constants on such small and irregular samples are out of the question, but comparative results are readily obtained by this method. Description and tests of the apparatus will be found in the preceding paper.

### *Experimental Results*

In Table 1 are given the magnetic attractions in milligrams per gram of oxide at the maximum magnetizing current of 800 milliamp., for each of the subdivisions of the section of the crystal. The letters correspond to those in Fig. 1. The attractions at the other values of magnetizing current would give no additional information for our present purpose, and are omitted.

Some idea of the susceptibility in absolute measure of the less magnetic portions may be obtained from the fact that ferrous ammonium sulphate, $FeSO_4.(NH_4)_2SO_4.6H_2O$, under the same conditions, gives an attraction of 2.68 mg. per gram.

The percentage of ferrous oxide, FeO, as determined by the chemical analysis of several of the portions of which there was a sufficient quantity available, is given in the fourth column of the table. The method of analysis was the same as that used in previous work on the oxides of iron, and has been described elsewhere.[3]

[3] J. C. Hostetter: *Journal of the American Chemical Society* (1916), **38**, 821.

The data for those portions that were tested both magnetically and chemically are plotted in Fig. 2. The figure also contains a curve showing the relation between the magnetic attraction at 800 milliamp and the percentage of FeO in mixtures of magnetite and quartz.

The magnetic data, supported by the chemical analyses, show that the hematite crystal from Elba varies in composition from the outside

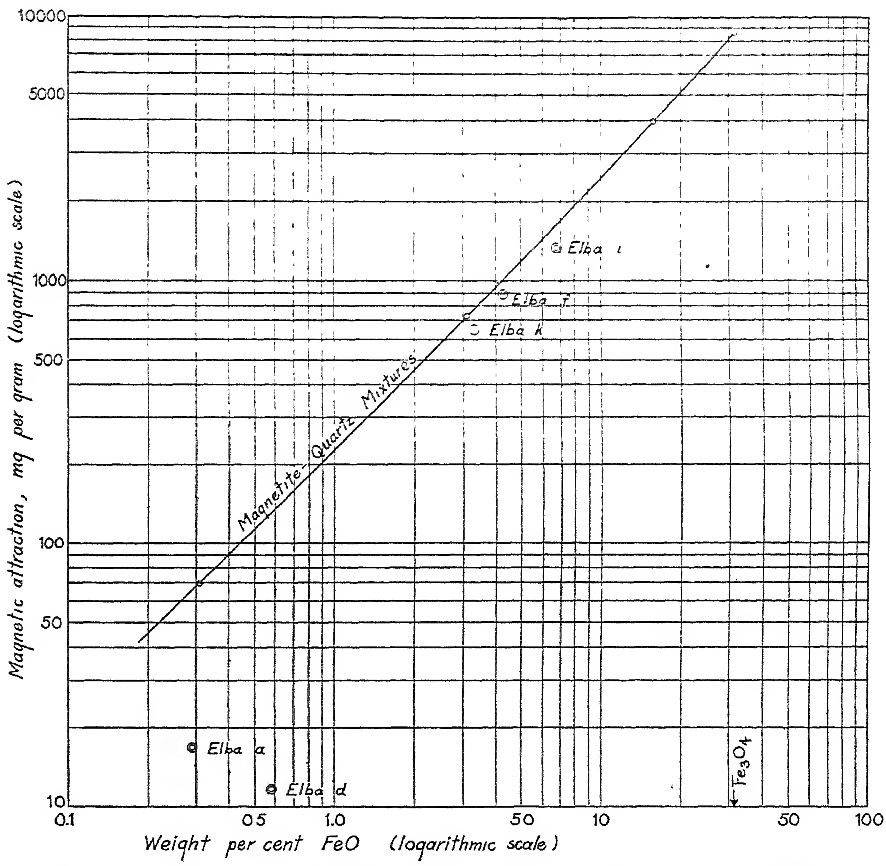


FIG. 2.—RELATION OF MAGNETIC ATTRACTION TO PERCENTAGE OF FeO.

to the center, and, in particular, from the "tip," which is bounded by good faces and evidently grew freely without interference, toward the "base," which is irregular as if broken out of the wall of a cavity. This variation of composition is shown graphically in Fig. 3, in which "contours" are drawn in the plane of the section, each line representing in a general way the locus of points of the same magnetic susceptibility, and hence of the same composition. The dotted lines are drawn to indicate the

probable extension of these lines of constant composition from the base of the crystal into the matrix of iron ore from which it was broken.[4]

TABLE 1.—*Magnetic Attraction on Various Portions of Elba Hematite Crystal*

| Portion | Weight of Sample Tested | Attraction at 800 Milliamp | Percentage of FeO |
|---|---|---|---|
| | Mg | Mg per Gram | |
| *a* | 634.4 | 16 8 | 0 29 |
| *b* | 185.5 | 17 6 | |
| *c* | 325.1 | 14 2 | |
| *d* | 328.1 | 11 7 | 0.58 |
| *e* | 508.2 | 23 3 | |
| *h* | 493.1 | 80 9 | |
| *g* | 238.9 | 75.9 | |
| *j* | 230.6 | 138 4 | |
| *l* | 349.5 | 222 4 | |
| *f* | 483 3 | 897. | 4.24 |
| *i* | 578 4 | 1,318 | 6.76 |
| | 577.1 | 1,327 * | |
| *k* | 847.8 | 654. | 3.31 |
| *m* | 126.3 | 874. | |

* Redetermination after removing and mixing the powder, to show degree of reproducibility of measurements on the most ferromagnetic member of the series.

## CAUSE OF THE ZONAL GROWTH

It is of interest to inquire what physical and chemical conditions could have produced such a zonal growth.

The origin of the Elba iron ores was for many years in controversy. Vom Rath, Cocchi, and Reyer, among the earlier geologists, believed the deposits to have been originally sedimentary. Later work, especially that of Lotti,[5] has shown that the ores are associated with post-Eocene salic igneous intrusions, and that they belong to the class of contact deposits.

The precise nature of the process by which these and other similar deposits were formed is of course difficult or impossible to state on the

[4] Measurements of the relation of physical properties, such as elasticity and magnetic susceptibility, to the crystal axes have frequently been made on large hematite crystals, usually on the assumption that the crystals were homogeneous. The present results show that the possible presence of zonal growth should always be taken into account in such measurements before conclusions are drawn as to the relation of properties to crystal axes. This is especially true of magnetic properties, on which a small percentage of FeO has so large an effect.

[5] B. Lotti: Descrizione geologica dell'Isola d'Elba. *Mem. descr. carta geol. Italia* (1886), 2, 193–235.

basis of field evidence alone. The explanation must be given in such general terms that no quantitative deduction can be drawn. The Italian mining geologist Cortese makes the following statement concerning the ores: "They are due to endogenous emanations of fluids and vapors at high temperature and under great pressure, which bore along with them the oxides of iron that were ultimately deposited in the hollows of the older rocks."

Let us put this statement into more precise terms, in order to find what deductions can be made from it and how they agree with the facts. Let us assume, for example, that the ores were deposited from a vapor which escaped from the igneous rock, and which contained iron, chlorine, hydrogen, and oxygen, at a temperature somewhere between 100° and 700°, and a pressure of 300 atmospheres or less. Now a mixture of the compounds of iron, chlorine, oxygen and hydrogen under these conditions is a

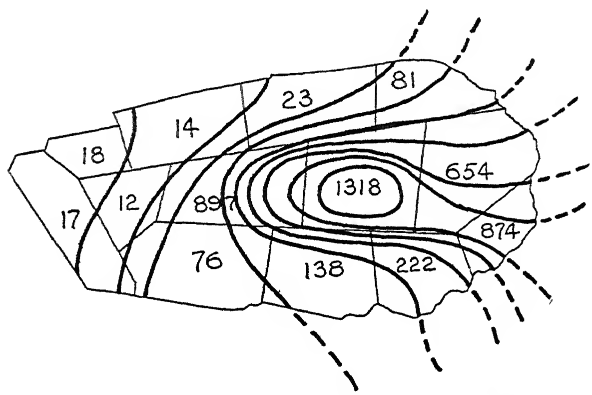


FIG. 3.—DISTRIBUTION OF FeO IN CROSS-SECTION OF ELBA CRYSTAL.

four-component chemical system. The phase rule and the chemical properties of the system enable us to draw certain conclusions concerning its behavior.

Two solid oxides, both being members of the hematite-magnetite series, cannot deposit out of such a vapor simultaneously, because the oxygen pressure falls continuously throughout the series $Fe_2O_3$-$Fe_3O_4$. Since the oxygen pressure in the vapor has a definite value, there can be only one oxide of this series in equilibrium with the vapor.

As only one oxide is depositing, the system consists of two phases: solid and vapor. If the original composition of the vapor is fixed, then the ratio of chlorine to hydrogen in the vapor must remain constant, since only iron and oxygen are depositing. The system is therefore tervariant; that is to say, there may be a range of pressures, a range of temperatures, and a range of concentrations of one of the components with

respect to the others, within which the deposition of the oxide may be taking place.

Disregarding for the present the doubled molecules $Fe_2Cl_6$ and $Fe_2Cl_4$ (which dissociate as the temperature rises), we may consider the deposition of oxides from the supposed vapor as based upon the following reactions:

$$2FeCl_3 + 3H_2O \rightleftarrows Fe_2O_3 \text{ (solid)} + 6HCl \tag{1}$$

$$FeCl_2 + H_2O \rightleftarrows FeO \text{ (solid)} + 2HCl \tag{2}$$

As we have shown experimentally, FeO and $Fe_2O_3$ form a solid solution from $Fe_2O_3$ toward $Fe_3O_4$. This solid solution acts as a single solid phase. The reactions (1) and (2) can therefore occur simultaneously, and yet, as shown above, a single oxide phase of varying composition can be depositing through a range of temperatures, a range of pressures, or a range of concentrations. Under conditions of complete equilibrium, the composition of this oxide should change as a whole, as the pressure, temperature, or composition of the vapor changed. Actually, however, the interior of crystals already deposited might be protected against further reaction by the continued deposition of more oxide on the surface. The result would be a zoned crystal like the Elba crystal which we have just described. Similar zonal growths are very common in minerals which form solid solution series, as, for instance, in the lime-soda feldspars.

We have assumed, for the sake of definiteness in discussion, that the iron was carried in a vapor containing iron, chlorine, hydrogen, and oxygen. It is not, however, necessary for the production of the zonal crystals that they should have been deposited from this particular vapor, or, in fact, from any vapor. The general principles remain the same, even though the iron were carried in a vapor of some other composition, or in a liquid solution.

### *Conditions Indicated by the Zonal Growth*

Two possible conditions of growth of the zoned crystals must be distinguished at this point:

1. The crystals may have formed in an open fissure through which vapor or solution was continually passing, supplied from a distant source.

2. The crystals may have grown in a closed space to which vapor or solution was not continuously supplied.

In either case, as we have shown above, a varying temperature, pressure, and concentration of the vapor is compatible with the deposition of the mixed oxide. But the phase rule gives no information as to *how* the percentage of ferrous iron in the crystal will vary with a given change in pressure, temperature, or concentration. A complete quantitative statement can only be made if we know (1) the equilibrium constants

of the reactions that take place, (2) the heat capacities and heats of formation of the compounds that take part in the reactions, and (3) the volume changes that take place in the reactions. This information is available only in part. We have good data,[6] for instance, on the reaction

$$4HCl + O_2 \rightleftarrows 2H_2O + 2Cl_2 \qquad (3)$$

which is one of the reactions that would occur in the vapor postulated above. Other reactions which would occur, and whose constants determine the equilibrium, are:

$$2FeCl_3 \rightleftarrows 2FeCl_2 + Cl_2 \qquad (4)$$

$$H_2 + Cl_2 \leftrightarrows 2HCl \qquad (5)$$

Until further experimental data on the reactions (1) to (5) are obtained, it will not be possible to make quantitative deductions regarding the effect of the different variables upon the ferrous-ferric ratio in the solid oxide. Certain qualitative deductions are, however, possible. The apparently continuous gradation of ferrous iron in the zoned crystal indicates that there was a continuous change in the temperature, pressure, and concentration relations of the vapor. If the crystals grew in a closed space, cut off from fresh supplies, it appears probable that falling temperature and the removal of ferrous and ferric iron from solution would determine the equilibrium. If they grew in an open fissure, continually supplied by vapor or solution from the igneous source, the zoning might represent (1) decreasing temperature of the source, (2) decreasing pressure in the supply, or (3) changing composition of the vapor or solution as the supply of constituents capable of escaping became exhausted.

It is an interesting fact that the zoning of the crystals is duplicated in the orebodies themselves. In general, the magnetite-bearing ore and the specularite (which always carries ferrous iron) are found in the deeper parts of the deposits, while the compact hematite is found in the central and upper portions. The hematite frequently passes into limonite in the upper portion, especially where the ores terminate against limestone; whether this limonite is the product of hypogene or supergene solutions must remain for the present an open question. The fact remains that there is a decrease in ferrous iron in passing upward through the orebodies, a condition that may well have resulted from the same series of changes in temperature, pressure and concentration as those which gave rise to the zoned crystals.

This association of the oxides is not peculiar to the Elba ores, but has been reported in other contact deposits. The more ferrous oxide, magnetite, is found nearer the igneous source, while the more ferric

[6] G. N. Lewis: Equilibrium in the Deacon Process. *Journal of the American Chemical Society* (1906), **28**, 1380–1395.

oxide, hematite, is found farther away. At Cornwall, Pa., for instance, "we have a foot wall of diabase, a hanging wall of limestone, with the [magnetite] ore between. . . . . We also find a small amount of hematite mostly at or near the contact with the limestone."[7] In the Firmeza district, Cuba (Juragua Iron Co. mines), "as the ore becomes less massive toward the margin of the deposit the proportion of specularite to magnetite increases."[8] "There is an increase in hematite in the higher levels of the mines."[9] In the Daiquiri district of southern Cuba, also, "hematite appears to be more abundant close to the surface than in depth."[10] "The fact that there is this definite arrangement of the increase of hematite above and increase of magnetite in depth is too consistent and widespread to be fortuitous when you consider that the Daiquiri mines at which Dr. Lindgren observed it, and the Juragua and the Ocania mines, stretch over a distance of about 15 miles, and in each case the same arrangement seems to hold good."[11]

Lindgren and Ross interpret the distribution of oxides in the Cuban deposits as evidence of the oxidation of a magnetite ore by oxygen-bearing waters from the surface. Roesler, Graton, and others believe that the distribution is either original or produced by the action of heated solutions from below upon the original oxide. In the preceding article we have noted that the magnetic properties of our specimen of the Juragua ore are not in accord with Lindgren's interpretation. It is only necessary to add that, as we have shown above, the zonal distribution of ferrous iron in such a deposit is readily accounted for by the same considerations as to changing temperature, pressure, and concentration of a vapor or solution as were used to account for the zonal distribution of ferrous iron in the Elba crystal. But we should not exclude the possibility that some surface oxidation has been superposed on the previously existing ferric percentage, and in addition it must be remembered that oxidation of pyrite may have produced a little additional ferric oxide in the form of limonite.

The possible action of an acid vapor or solution, such as our assumed iron-chlorine-oxygen-hydrogen vapor, upon calcium carbonate is of special interest in connection with the Elba deposits. The ores terminate in many places against limestone, and have evidently replaced large bodies of this rock. This is shown clearly by the residual blocks of limestone found in unaltered stratigraphic position within the ore,[12] and by the

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[7] H. Souder: Discussion in *Trans.* (1916), **53**, 66.

[8] M. Roesler: *Trans.* (1916), **56**, 99.

[9] M. Roesler: *Op. cit.*, 103.

[10] W. Lindgren and C. P. Ross: *Trans.* (1916), **53**, 55.

[11] M. Roesler: Discussion in *Trans.* (1916), **53**, 66.

[12] B. Lotti: *Geol. För. Förh.* (1891), **13**, 599–603.

fact that even the original caverns of the limestone have sometimes been preserved in the limonitic portions of the orebodies.[13]

The principal reaction occurring at the contact of a vapor, such as we have postulated, with the limestone would be:

$$CaCO_3 + 2HCl = CaCl_2 + H_2O + CO_2.$$

From reactions (1) and (2) it is evident that the removal of HCl from the vapor by reaction with $CaCO_3$ would cause the deposition of oxides of iron, while the $CaCl_2$ formed would pass off in solution. The lime of the $CaCl_2$ would probably not be redeposited elsewhere, on account of the great solubility of $CaCl_2$ and the absence of any precipitating agent. The $CO_2$ would mostly escape as gas, unless water was in large excess, in which case considerable lime would be taken into solution as bicarbonate. According to Capacci,[14] neither gypsum (calcium sulphate) nor travertine (calcium carbonate) are found in association with the Elba ores; he believes that the former fact stands in the way of any theory involving sulphate solutions, while the latter suggests the absence of large amounts of carbonated waters.

A suggestion as to the conditions under which specular crystals such as those from Elba may be deposited can be obtained from some excellently developed artificial hematite crystals which were kindly furnished us by Prof. Munroe of George Washington University. He obtained them some years ago from a Deacon-process apparatus in Philadelphia.[15] The Deacon process depends upon the oxidation of hydrochloric acid by atmospheric oxygen, and is usually operated at a temperature of about 400°. The crystals in question were taken from the inside of the digester pipes, after a shutdown which was due, not to corrosion from within, but to the burning through of the pipes by $SO_3$-bearing furnace gases from without. The iron of the hematite crystals may have come in as an impurity in the hydrochloric acid, and was probably deposited according to reactions (1) and (2), page 440. We found by analysis that these crystals contain 0.21 per cent. FeO, which is about the same as the tip of the zoned Elba crystal. It should be remembered, however, that the ferrous-iron-bearing Deacon-process crystals were deposited out of a strongly oxidizing vapor; the tip of the Elba crystal might have grown at a very different temperature and pressure if the composition of the vapor from which it deposited was very different from that in the Deacon process.

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[13] C. Capacci: The Iron Mines of the Island of Elba. *Journal of the Iron and Steel Institute* (1911), **84**, 412–450.

[14] C. Capacci: *Op. cit.*, 442.

[15] C. E. Munroe: Artificial Hematite Crystals. *American Journal of Science* (1907), **24**, 485–486.

## Summary

The powdered oxide from certain crystals of hematite from Elba contains considerable FeO, and can also be fractionated magnetically. It is therefore not homogeneous, as would be the case if the crystal were a uniform solid solution throughout. Analyses and magnetic measurements on a cross-section of an Elba crystal showed that the magnetic susceptibility and percentage of FeO vary, not irregularly, but continuously, being highest at the base and lowest at the free-growing tip of the crystal. The crystal is therefore zoned with respect to its FeO content.

Since $Fe_3O_4$ goes into solid solution in $Fe_2O_3$, forming a single solid phase of varying composition and properties, a zonal distribution of FeO is to be expected in an oxide of iron depositing from a vapor or solution. The occurrence of such zonal growth indicates continuously changing conditions of temperature, pressure and concentration during the formation of the crystals. Several ore deposits of contact-metamorphic origin show a zonal distribution of ferrous iron, probably arising from the same causes as the zoning of the single crystals.

# The Tayeh Iron-ore Deposits

BY CHUNG YU WANG, HANKOW, CHINA

(St. Louis Meeting, September, 1917)

DURING the time I was in charge of this mine, from 1914 to 1915, I had occasion to read the interesting papers by T. T. Read and C. M. Weld about these deposits, to find how far their observations corroborated mine. To quote from the paper on "The Mineral Production and Resources of China" by T. T. Read:[1] "The iron ores at Tayeh * * * * lie along the contact between a marble and an intrusive body of a dark gray syenitic rock. * * * * At one place it is slightly magnetic, apparently having been partly reduced to the magnetic oxide by the action of reducing solutions, which have deposited small amounts of copper and iron sulphides along the foot wall." To quote from the paper on "The Tayeh Iron Ore Deposits"[2] by C. M. Weld: "I find the orebodies described as occurring along an approximately E-W contact between hornblende-granite and limestone. * * * * The granite is a granitoid rock consisting essentially of quartz, potash-feldspar and hornblende." These quotations show the differences of opinion entertained by two observers. My own study of these interesting deposits has been intermittent amid the manifold duties devolving upon a superintendent.

## General Description of the Deposits

The deposits are situated to the west of the Yangtse River, being connected with the river by a standard-gage railroad of 56 li (16.16 miles, 26 km.) in length, as shown in Fig. 1. The general topography of the region is of mature age, showing broad valleys with sluggishly meandering streams. These deposits are worked as two mines, named the Tieh Shan Mining Department and the Teh Tao Wan Mining Department; the former consisting of three orebodies more or less connected, named Tieh Man Kan, Sha Mau Tze and Lung Tung, and the latter consisting of Chang Pei Shan, Sze Tze Shan, Ta Shih Men, Yeh Chih Ping and Tsin Shan, also more or less connected, as shown in Fig. 2.

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[1] *Trans.* (1912), **43**, 29–30.
[2] *Trans.* (1912), **44**, 27–37.

## The Country Rock

The deposits lie along the contact between an igneous intrusion, belonging to the hornblende-syenite group—the syenite of Read, diorite of Seltzer, and hornblende-granite of Weld—and limestone. The hornblende-syenite, invariably in a decomposed condition along the whole length of the contact, presents the appearance of a decomposed alkaline aluminum silicate rock of a light brownish color, sometimes, though

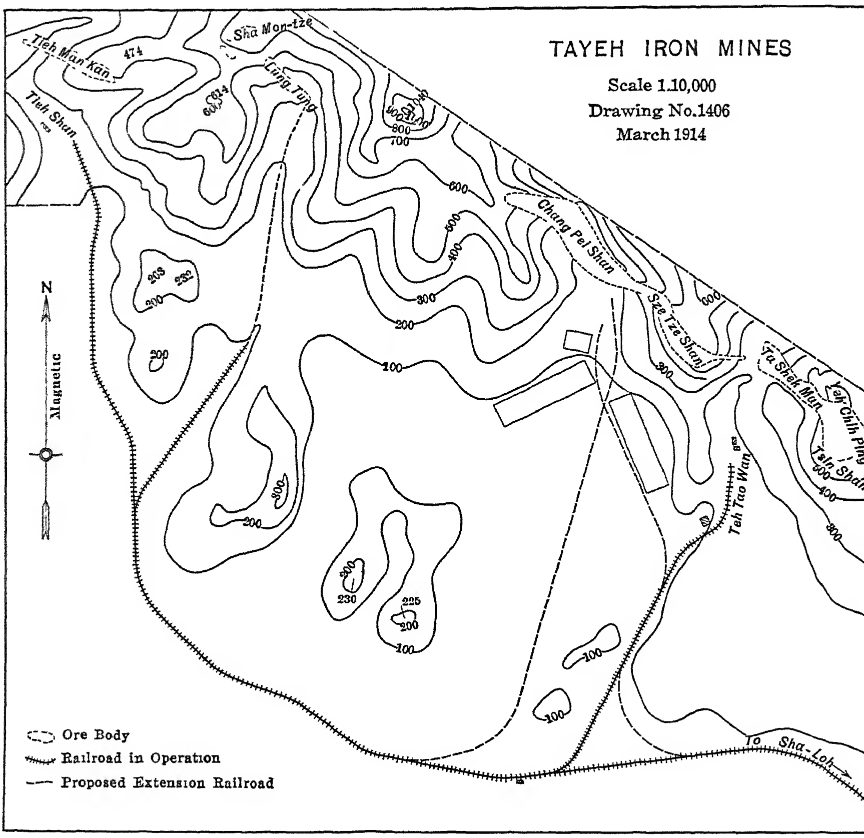


Fig. 1.

rarely, with a greenish tint to it, which is due to the greenish color of the partially decomposed hornblende. The essential minerals forming this rock, being medium-grained and of inequigranular fabric, are orthoclase and hornblende. The various stages of decomposition of the orthoclase mineral—from the fresh colorless crystals to a dull whitish kaolin product—can be clearly discerned with a field lens. It must be remarked here that there are two varieties of hornblende-syenite—the one, dark gray,

near the partially decomposed rock next to the orebodies; the other, mottled white, which is farther off from the deposits and can be found as boulders and pebbles along the streams. The limestone occurs in various stages of crystallization, in which hardly any trace of contact minerals can be found. I am indebted to my assistant, W. A. Wong, for having found a layer of limestone thickly studded with crystals of garnet, not far from the orebody; the only layer of limestone with such contact mineral that I saw during my stay in Tayeh.

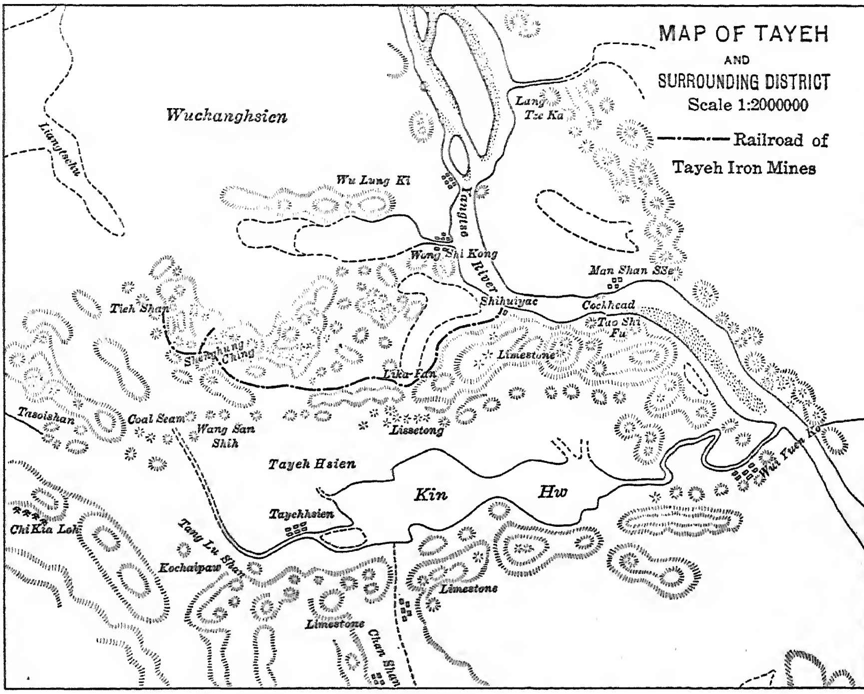


FIG. 2.

## DESCRIPTION OF THE DEPOSITS

From Fig. 2 it will be noticed that the deposits are more or less lens-shaped, having an approximate NW-SE strike and a NE dip of 60° to 70°, with the exception of the orebodies at Sha Mau Tze and Tsin Shan, which dip away from the igneous rock (SW dip); that is to say, the hornblende syenite forms the hanging wall, which is generally sharply distinct from the orebodies themselves, and the limestone the foot wall, which in many places is very irregular in outline, inclosing bunchy pieces of limestone, ranging from 2 to 10 ft. in diameter. The orebody at Tieh Man Kan has been faulted from the main trend of the other orebodies in a southwesterly direction. The iron ore is essentially composed of a mix-

ture of magnetite and hematite, with occasional occurrences of malachite, chalcocite and pyrite and rarely epidote and mica. Table 1 shows average analyses of the ore and of the limestone.

TABLE 1.—*Average Analyses of Ore and Limestone*

Ore:

| No. | Localities | For | Fe, Per Cent. | Mn, Per Cent. | S, Per Cent | P, Per Cent. | $Al_2O_3$, Per Cent. | Cu, Per Cent. | $SiO_2$, Per Cent. | Sp. Gr. |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Teh Tao Wan | Wakamatsu & Hanyang | 62.93 | ... | 0.029 | 0.049 | .... | 0.238 | 5.42 | 4.64 |
| 2 | Tieh Shan | Hanyang | 59.45 | 0.63 | 0.319 | 0.109 | 1.05 | 0.292 | 9.16 | 4.13 |

Limestone:

| $SiO_2$, Per Cent. | $Al_2O_3Fe_2O_3$, Per Cent. | CaO, Per Cent. | MgO, Per Cent. | Sp. Gr. |
|---|---|---|---|---|
| 3.14 | 1.14 | 52 63 | 0.92 | 2.71 |

Analyses of the discarded ore dumped into the waste heap have been found to contain as much as from 1 to 2 per cent. S and from 0.5 to 3 per cent. Cu. All the orebodies are more or less jointed and the jointing planes are in a general way parallel to the hanging wall, along which is frequently found copper carbonate which must have been deposited and concentrated there from the leaching of the chalcocite found disseminated in certain portions of the orebodies.

## Origin of the Deposits

I consider that these orebodies belong to the contact-deposit type and are the products of contact metamorphism; that the magnetite is primary—it has been in places partially or wholly altered to hematite, due to hydrothermal and pneumatolytic reactions, concomitant with the intrusion of the hornblende-syenite—and that the hematite thus originated has been carried forward by ascending telluric water, probably together with the aid of descending meteoric water, to replace metasomatically the intrusive and the limestone, especially along the contacts at certain particular spots. I have arrived at this conclusion from a consideration of the following facts:

1. As noticed above, the hornblende syenite along nearly the whole length of the contact is in a decomposed and altered condition, due undoubtedly to hydrothermal- and hydro-metamorphism, that involves the development of kaolin and chlorite with traces of epidote, pyrite, quartz and calcite. This may be the result of the process of propylitization.

2. At Ta Shek Man, the hornblende-syenite changes to a light-brownish and yellowish color and is rather soft, with scales of sericite, distin-

guishable megascopically. This is probably the result of sericitization, developing prominently in this particular locality.

3. The crystallization of the limestone to marble is undoubtedly due to its contact with the hornblende-syenite magma, with the development of a banded structure in the limestone, particularly discernible at Sha Mou Tze. This banded structure may probably be due to differing degrees of silicification.

4. Near the main orebodies at Teh Man Kan, irregular smaller bodies of ore rich in manganese are found in the limestone, which must owe their origin to the same magmatic emanation.

5. I regard the magnetite as primary and derived from magmatic emanation rich in iron, and the hematite as a secondary product from the

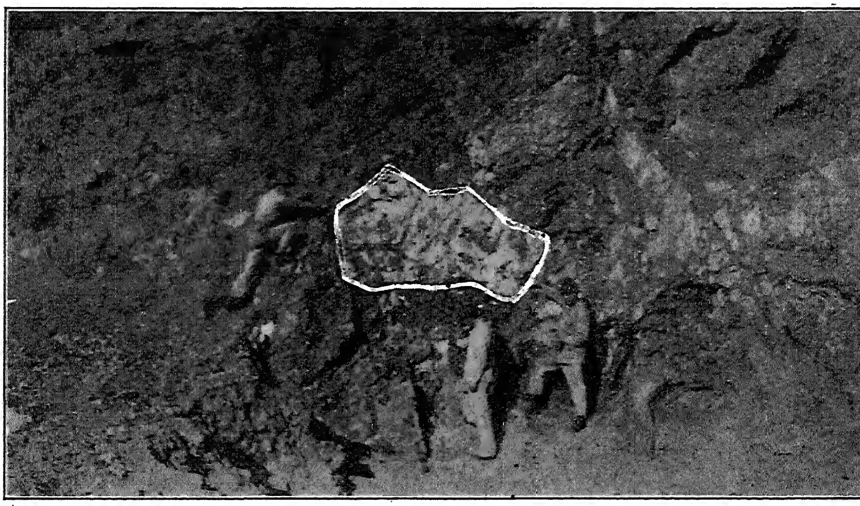

FIG. 3.—DECOMPOSED HORNBLENDE SYENITE TOTALLY INCLOSED BY HEMATITE. DEPOSIT AT YEH CHIH PING.

oxidation of the magnetite. We can picture to ourselves how this process of hematization can be brought about. Chemically magnetite is composed of the sesquioxide $Fe_2O_3$ which is hematite, and the protoxide FeO, which, being unstable, would readily be oxidized to $Fe_2O_3$ in the presence of active free oxygen under suitable pressure and temperature. Undoubtedly the gaseous vapor, during the intrusion of the syenite, furnished such oxygen. O. Silvestri has found as much as 18.97 per cent. of oxygen in the gas issuing from Mt. Etna. It is generally known that chemical reaction may or may not be favored by an increase of pressure, and this is why magnetite cannot be hematized either at or near the surface, or at great depth. Probably the gaseous emanation rich in magnetite and oxygen, on coming up from below, reaches a particular zone where

the pressure and temperature are suitable to the combination of the oxygen and magnetite to form hematite.

6. Some of the hematite thus produced may be carried forward by telluric water, probably aided by descending meteoric water to replace metasomatically some portion of the rocks of both the hanging wall and the foot wall. This process of replacement is clearly shown from Figs. 3 and 4 where decomposed hornblende syenite is totally inclosed by hematite; from the occurrence of big bunchy pieces of limestone, as de-

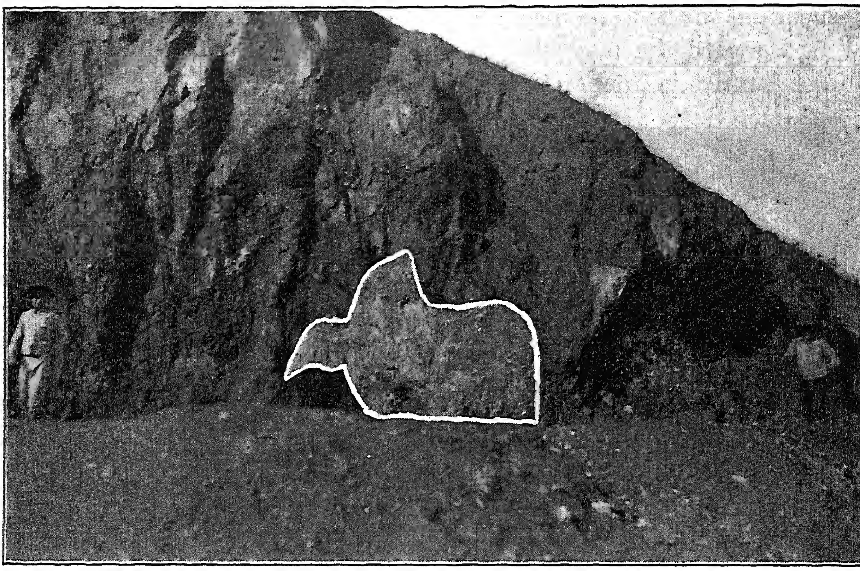

Fig. 4.—Formation at Tsin Shan of same nature as that shown in Fig. 3.

scribed above, inclosed entirely by hematite, and from some hand specimens in which particles of altered syenite are in process of being replaced by hematite.

## Production

The following table shows the output of iron ore from 1896 to 1914 inclusive, which, in round numbers, is about 3½ million tons. The ore-in-sight of these deposits, including Chang Pei Shan, which, being Government property, has not yet been opened, has been estimated by three engineers to be 20, 30, and 100 million tons respectively. But according to my judgment, it would not be unreasonable to regard the ore-in-sight as 30 million tons and the probable-ore in depth as 30 million tons.

| Year | Tons |
|---|---|
| 1896 | 15,933 |
| 1897 | 20,545 |
| 1898 | 36,558 |
| 1899 | 24,765 |
| 1900 | 57,201 |
| 1901 | 109,215 |
| 1902 | 84,036 |
| 1903 | 107,794 |
| 1904 | 106,378 |
| 1905 | 151,168 |
| 1906 | 185,610 |
| 1907 | 174,612 |
| 1908 | 171,934 |
| 1909 | 309,399 |
| 1910 | 343,097 |
| 1911 | 359,467 |
| 1912 | 268,685 |
| 1913 | 416,342 |
| 1914 | 488,258 |
| | 3,430,976 |

## Discussion

Thomas T. Read, Palmerton, Pa. (written discussion*).—It is much to be regretted that in presenting a further discussion of these deposits, first described by myself and later discussed by C. M. Weld, Mr. Wang has not made a more complete presentation of the evidence on which he bases his conclusions as to their origin. Mr. Weld said in his paper that his visit was confined to a few hours, that he saw only one of the several exposures, and that he took away no samples for further study. My own visit was brief, and for another purpose than the study of the geological features of the deposits, so the fact that Mr. Weld described as a hornblende-granite the rock which I described as a "dark gray syenitic rock," and which Mr. Wang identifies as a hornblende-syenite, seems of no great importance, since this rock is generally decomposed along the contact, presumably the only place in which Mr. Weld observed it. Later in the same paper (p. 31), Mr. Weld says that his opinion is that the magnetite of the deposits is primary, but Mr. Wang makes no mention of this in stating his own conclusions to the same effect. When Mr. Weld's paper appeared, I did not discuss it, because, although I considered the evidence that he brought forward to support his belief that the magnetite is primary too slender to support the hypothesis, my own evidence to the contrary was equally tenuous. The opportunities for observation and study open to Mr. Wang must have provided him with

* Received Apr. 2, 1917.

abundant evidence as to the origin of the deposits, and it is much to be regretted that he has not seen fit to include it in his paper, instead of making a brief statement of his conclusions.

Before accepting the statement that these deposits are of contact-metamorphic origin, the magnetite being primary, it is proper to inquire how closely they resemble the deposits elsewhere which have been ascribed to contact-metamorphic action. In the Banat of southern Hungary, there are iron deposits which various European geologists are agreed are due to the eruption of syenitic rock through the Tithonian limestone. The iron ore is magnetite, but it is accompanied by garnet, wollastonite, vesuvianite, and other common products of contact metamorphism. In Russia, at Wyssokaia-Gora, in the Nijne-Tagiulsk district, syenite has intruded Devonian limestones. Garnet-epidote rock is abundant, and is generally considered to be the product of the contact metamorphism. The iron deposits of the Island of Elba are invariably accompanied by garnet and epidote. The deposits at Traversella and Brosso, in the Piedmont, Italy, are famous for their garnet crystals. At Dannemora, in Sweden, the limestone shows characteristic garnet. It would seem, therefore, that any complete discussion of the origin of the Tayeh deposit must include an explanation why, to quote Mr. Wang, "hardly any trace of contact minerals can be found." It is also difficult to reconcile the presence of "chalcocite found disseminated in certain portions of the orebodies" with an oxidizing action sufficiently vigorous to convert magnetite into hematite. I do not deny that these deposits may be due to contact action, but the evidence so far presented to support the hypothesis appears to be too incomplete to warrant its unreserved acceptance.

# Manganiferous Iron Ores of the Cuyuna District, Minnesota*

BY E. C. HARDER,† PH. D., WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

## Introduction

In view of the gradually decreasing known reserves of high-grade manganese ore and the rapidly increasing consumption of iron-manganese alloys in the steel industry, it is well to turn our attention to the development of low-grade manganiferous ore deposits and to the utilization of these ores in steel manufacture. The dependence of this country on imported manganese ore and ferromanganese has been forcibly brought to the attention of steel makers during the last few years by the cessation of manganese-ore imports from Russia, the restriction of manganese-ore imports from India and of ferromanganese imports from Europe. Should the importation of manganese ore from Brazil cease, the steel industry of this country would face a serious situation. The present practice of steel making requires ferromanganese in large quantities and ferromanganese of the grade used at present can be produced only from high-grade manganese ores such as the United States appears to possess only in insignificant amounts. There are in the United States, however, large reserves of manganiferous iron ore which could probably be used extensively in the steel industry if certain changes in the practice of steel manufacture were made. Thus high-grade manganese ore could be conserved. Among the most important deposits of manganiferous iron ore in this country are those of the Cuyuna iron-ore district of Minnesota. It is probable that in case of restriction of foreign supplies, the users of manganese in the United States would have to rely to a large extent on the Cuyuna district to make up the shortage.

Manganiferous iron ore was found in association with iron ore in the Cuyuna district in many of the early drilling operations and the possibility of its utilization was in the minds of various explorers in the dis-

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† Geologist, U. S. Geological Survey.

trict for a long time before mining operations were begun. Many drill holes in the northern part of the district passed through iron-bearing formation containing varying amounts of manganese. The distribution of the manganese was found to be very irregular, however. In certain drill holes high-grade manganiferous iron ore, containing 25 per cent. manganese and over, was encountered, while nearby holes showed little or no manganese. Even in the same hole great variations were found in the manganese content of the ore. Ore with more than 20 per cent. of manganese was found to be irregularly intermixed with material containing 2 or 3 per cent. of manganese or even less. Generally the iron content was found to be high when the manganese content was low and *vice versa,* so that the combined iron and manganese content was fairly regular. In many drill holes, however, material was encountered which was abnormally high in silica and low in combined iron and manganese.

In view of this situation and of the lack of demand for manganiferous iron ore in the industries, the mining of these ores was discouraging for a long time, and iron ores containing an appreciable amount of manganese were considered almost worthless. In 1911, however, the shaft of the Cuyuna-Mille Lacs Iron Co. was sunk and in 1913 the first manganiferous iron ore produced in the district was shipped from this mine. In 1913 a second shaft, that of the Iron Mountain Mining Co., was sunk and a few carloads were shipped from this mine for experimental purposes during the same year. The production gradually increased in succeeding years and as the supply of manganese in the country diminished due to war conditions, and its price rose, the demand for Cuyuna Range manganiferous iron ores increased. During the coming season, such ores will probably be produced from 14 or 15 mines in the district.

While this increase of demand for Cuyuna Range manganiferous iron ore has been produced largely under abnormal conditions, it seems probable that its extensive use in the industries will result in certain changes of metallurgical practice and it is a live question whether on account of such changes the demand for these ores will continue when normal conditions are reëstablished.

## Situation of the Cuyuna District

The Cuyuna iron-ore district is situated in the central part of Minnesota, in Aitkin, Crow Wing, Morrison, and Todd Counties. The main central portion of the district is 90 miles (145 km.) west of Duluth and 55 miles (88.5 km.) southwest of the western part of the Mesabi district. The productive area lies south of the Mississippi River in the central and southern part of Crow Wing County, but extensions of the iron-bearing beds are known to the northeast as far as the central part of

Aitkin County and to the southwest in Morrison County and in the eastern part of Todd County. In the part of Crow Wing County north of the Mississippi River, also, exploration work has shown the existence of iron-bearing rocks at several localities.

The productive part of the Cuyuna district is commonly divided into two sections known as the north and south ranges. The south range includes the portion of the district lying south of the Duluth-Brainerd branch of the Northern Pacific Railway which runs southwestward through the district. The north range lies north of the railway. The two ranges, while exhibiting certain differences in the character of the

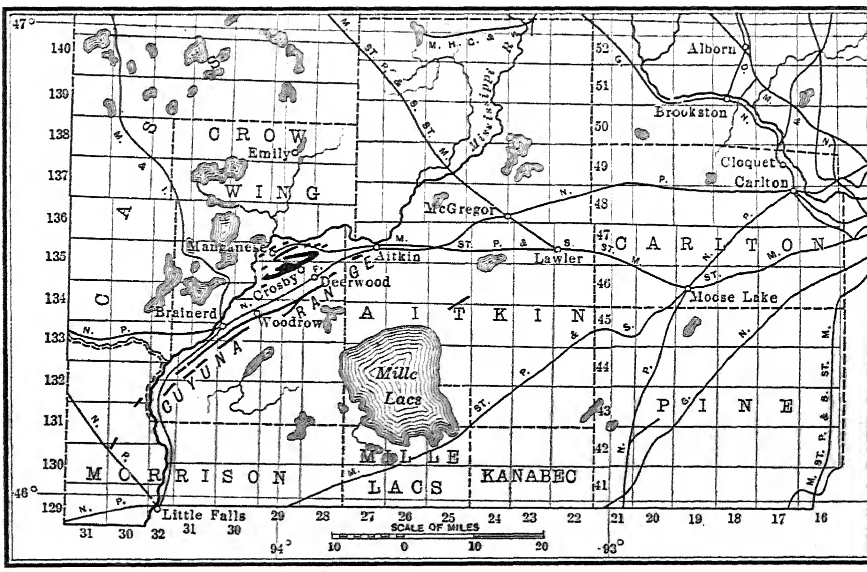


FIG. 1.—MAP SHOWING THE LOCATION OF THE CUYUNA IRON-ORE DISTRICT.

ore, are geologically and structurally similar. The south range with its extensions occupies a large stretch of territory. It has a known length northeast and southwest of about 65 miles (104.6 km.) and a maximum width of 3 or 4 miles (4 to 6 km.). Seven mines and exploration shafts have been sunk along it, of which four have produced ore, but of which only one, the Wilcox mine, has been in steady operation. Many undeveloped orebodies, the existence of which has been shown by drilling, are found along the south range. The north range occupies a comparatively small area, being about 10 miles long and 5 miles wide, but it contains most of the important mines in the district. It is in the northern part of the north range that the principal manganiferous iron-ore deposits

are found. There are at present 17 producing mines on the north range, while 7 or 8 new mines are being developed. Ten of the active mines have produced manganiferous iron ore. The north range is separated from the south range by a strip of territory from 2 to 3 miles wide in which iron-bearing rocks have not yet been found.

The iron-bearing rocks occurring in Crow Wing County north of Mississippi River are found mainly in the region of Emily and to the southeast between Emily and Aitkin. Ore has been shown to exist in this region, but no attempt has been made to mine it.

East central Minnesota is for the most part a low-lying region. The portion occupied by the Cuyuna district contains numerous low morainic hills interspersed with lakes and swamps. Extensive marshes occur in Aitkin County east and northeast of the district, while to the southwest and south are large outwash plains. The elevations in general range from 1150 to 1350 ft. (350 to 411 m.) above sea level, or from 548 to 748 ft. (167 to 227.9 m.) above the level of Lake Superior. Much of the area is under cultivation.

Two railroads serve the district, the Duluth-Brainerd branch of the Northern Pacific Railway and the Cuyuna Range branch of the Minneapolis, St. Paul and Sault Ste. Marie Railway. Up to the present time the latter has handled most of the iron-ore traffic.

## Geology[1]

The bedrock in the Cuyuna district is covered by a mantle of glacial drift which varies in thickness from 15 to 400 ft. (4.57 to 121.9 m.) Outcrops are found at only two places, (1) near Dam Lake and Long Lake

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[1] F. S. Adams: The Iron Formation of the Cuyuna Range. *Economic Geology* (1910), **5,** 729–740; and (1911), **6,** 60–70 and 156–180.

C. A. Cheney, Jr.: Structure of the Cuyuna Iron-Ore District of Minnesota. *Engineering and Mining Journal* (1915), **99,** 1113–1115.

E. C. Harder and A. W. Johnston: Notes on the Geology and Iron Ores of the Cuyuna District, Minnesota. *U. S. Geological Survey Bulletin* 660*A* (1917); Preliminary Report on the Geology of East Central Minnesota, Including the Cuyuna Iron-Ore District. *Minnesota Geological Survey Bulletin* (in press).

C. K. Leith: The Geology of the Cuyuna Iron Range. *Economic Geology* (1907), **2,** 145–152.

E. P. McCarty: Manganiferous Iron Ores of the Cuyuna Range. *Engineering and Mining Journal* (1915), **100,** 400–402.

C. R. Van Hise and C. K. Leith: The Geology of the Lake Superior Region. *U. S. Geological Survey Monograph* 52 (1911), 211 *et seq.*

Carl Zapffe: The Cuyuna Iron-Ore District of Minnesota. *Brainerd Tribune Supplement,* Sept. 2, 1910; Geology of the Cuyuna Iron-Ore District, Minnesota. *Mining World* (1911), **34,** 585–588.

Carl Zapffe and W. A. Barrows, Jr.: The Iron Ores of the South Range of the Cuyuna District, Minnesota (1912), **44,** 3–13.

in central Aitkin County south of the eastern end of the iron-bearing region and (2) at Randall in Morrison County near the southwestern end of the iron-bearing region. The outcrops at Dam Lake and Long Lake consist of quartzite and basic igneous rocks while those at Randall are composed of chloritic schist. Both have suffered metamorphism and deformation and are related to rocks which have been shown by drilling to occur in various parts of the Cuyuna district in association with the iron-bearing rocks.

The glacial drift in most parts of the district is sandy near the surface though locally there are areas of clay and loam. Swamps and marshes are numerous and in these localities the drift is overlain by beds of marl and peat. In the central part of the district several open-pit mines show good sections of the glacial drift. In these the prevailing succession from top to bottom is as follows:

1. Muck, peat and marl. Not everywhere present.

2. Yellow and gray sand. Forms main part of drift mantle. It varies in coarseness and locally contains thin pebble beds.

3. Gray stratified clay. Usually consists of fine, laminated clay and silt with boulders and pebbles which increase in abundance toward the lower part. It is called "hard pan" by the miners. Generally it rests on the bedrock surface but in a few places it is underlain by

4. Red clay with boulders.

Sand forms by far the most important part of the drift and locally comprises the entire thickness of glacial material. The gray clay is also widespread but shows great variations in thickness and distribution. The other phases are local. The average thickness of the glacial mantle over the main central part of the district varies from 50 to 80 ft. (15 to 24.4 m.)

The rocks of the Cuyuna district can be grouped under three classes; (1) metamorphosed sedimentary and igneous rocks interlayered with each other in beds and lenses, usually with steep dip due to close folding; (2) igneous rocks intruded into the metamorphosed rocks subsequent to their metamorphism and deformation; and (3) younger rocks which lie horizontally on the eroded surfaces of the preceding.

The age of the various rocks is not definitely known. The old metamorphosed rocks have been classed as upper Huronian on account of their general lithological similarity to and apparent areal continuation with the upper Huronian rocks of the Mesabi district.[2] They consist principally of schists and slates of various kinds associated with beds and lenses of iron-bearing formation. Locally quartzite has been found and at one or two places limestone has been reported to occur. Meta-

[2] C. R. Van Hise and C. K. Leith: The Geology of the Lake Superior Region. *U. S. Geological Survey Monograph* 52 (1911), 211 *et seq.*

morphosed igneous rocks, many of which have been rendered schistose but some of which retain the original textures, are common.

The schists are principally quartzose and argillaceous sericitic schists, but locally chloritic schist is very abundant. Gray, finely micaceous phyllite is also common. Of the slates two principal kinds have been met with in the drilling: (1) dark gray to black graphitic slate, and (2) laminated grayish-green or green slate. The slates and schists occupy the areas between the iron-bearing belts, of which seven or eight trend northeastward through the district. They form the wall rocks of the ore-bearing beds or are interlayered with the ore-bearing rocks. The types most commonly found near the iron-bearing formation are green chloritic schist and laminated green slate. Graphitic slate, however, is also a common associate of the iron-bearing rocks. Where these rocks occur in close proximity to the iron-bearing formation they are generally heavily impregnated with iron and are dark red in color. They are then designated as ferruginous slate, ferruginous schist or paint rock. The other classes of old metamorphosed rocks are somewhat less abundant than the slates and schists. Quartzite and limestone probably occur only as small local lenses. Metamorphosed igneous rocks, however, and beds and lenses of iron-bearing formation are common and widespread. Metamorphosed igneous rocks which have in part been rendered schistose but in which igneous textures are still closely shown are especially abundant along the south range. They border beds of iron-bearing rock or occur interlayered with iron-bearing formation and slate. Most of them now consist of feldspar and chlorite. They were probably originally basic or semi-basic in character. Green chloritic schist, common both in the north range and south range, is believed to be an igneous rock so highly metamorphosed that the original textures have disappeared.

The iron-bearing formation of the Cuyuna district consists in its upper oxidized portion of ferruginous chert, ferruginous slate, iron ore and manganiferous iron ore, and in its deeper, unoxidized portion of cherty and slaty iron carbonate rocks. Amphibole-magnetite rock and magnetitic slate formed by the metamorphism of the iron carbonate rocks are found locally in both the upper and lower portions locally. In certain parts of the district both the original and oxidized phases of the iron-bearing formation are manganese-bearing. The former in many places contains as much as 3 or 4 per cent. manganese and the latter 30 per cent. or more.

The iron-bearing rocks are found as beds and lenses interlayered with the schists, slates and associated rocks. They occur in a number of parallel belts, but it seems probable that this duplication is due largely to intense folding and that only one main horizon of iron-bearing rocks exists. Both above and below this main horizon, however, there are locally smaller and less continuous lenses of iron-bearing rock separated

from the main formation by schist and slate. The main formation itself also in many places contains interbedded schist and slate which split it up into several beds, thus making the general occurrence very irregular.

The depth of oxidation of the iron-bearing rocks varies greatly in different parts of the district. In places the unaltered rock has been encountered less than 100 ft. (30.5 m.) below the bedrock surface while elsewhere oxidized phases have been found at a depth of 650 ft. (198 m.) or more below bedrock surface. Iron ore of good grade is known at this depth, indicating that locally the action of surface waters has been very extensive. In general, it appears that on the north range the oxidized iron-bearing rocks extend to greater depths than they do on the south range.

While most of the ferruginous chert, ferruginous slate, and ore in the Cuyuna district is doubtless derived from original ferrous carbonate rocks by weathering processes, it is not impossible that some of these rocks may have been originally deposited in their present form. Such rocks, of course, do not have the same limitations as to depth as the oxidized rocks have. The slates, schists, and other rocks of the district also show the effect of weathering for considerable distances below the bedrock surface locally, but not to the same degree as the iron-bearing rocks.

## Structure of the Rocks

The rocks of the Lake Superior region have suffered folding at several periods in their history. As far as is known, however, the forces causing the deformation at the different periods have been applied in approximately the same directions, *i.e.*, northwest-southeast, so that the results as apparent in the rock structure are superimposed upon each other. The final outcome of these various deformational activities has been that all the pre-Cambrian rocks of the Lake Superior region have been folded into a great complex synclinorium trending approximately northeast-southwest.

The rocks of the Cuyuna district are folded into a complex series of northeast-southwest trending anticlines and synclines.[3] The dip of the limbs of the folds is usually vertical or very steep and may be either to the southeast or northwest, the former dip predominating. Over considerable areas close folding has resulted in producing an approximate parallelism of both limbs of a fold. This is probably the explanation of

[3] C. R. Van Hise and C. K. Leith: The Geology of the Lake Superior Regions. *U. S. Geological Survey Monograph* 52 (1911), 620 *et seq.*

Carl Zapffe: The Cuyuna Iron-Ore District of Minnesota. *Brainerd Tribune Supplement,* Sept. 2, 1910.

C. A. Cheney, Jr.: Structure of the Cuyuna Iron-Ore District of Minnesota. *Engineering and Mining Journal* (1915), **99,** 1113–1115.

the great predominance of southeast dips in the district, which are characteristic both in the north and south ranges. The pitch of the folds is usually very low and may be either to the northeast or to the southwest. Pitches in one or the other direction usually predominate in different parts of the district. Thus at the southwestern end of the north range the folds in general pitch to the northeast, while along the central part of the north range they pitch to the southwest. Along parts of the south range, the folds pitch so slightly that the crests of anticlines are practically horizontal for miles. Because of this low pitch of the folds the various rock layers appear on the erosion surface as approximately parallel northeast-southwest trending bands. Locally, where the pitch brings ore-bearing bands below the erosion surface, sharp turns occur and the layers double back on themselves. Elsewhere minor drag folds cause local irregularities in their trend.

On account of the lack of exposures and as yet insufficient drilling and underground development, it has not been possible to work out the details of the structure. Drag folds and other secondary structures point to the existence of several major and many minor folds in both the north range and the south range. Ultimately, by means of these structures, it may be possible to determine the relationship between the rocks in different parts of the district. The apparent absence of a definite stratigraphic succession over any considerable area, however, is a serious drawback in working out the geologic relations.

The general distribution and structure of the rocks in east central Minnesota indicates that the Cuyuna district is situated near the axis of the southwestern extension of the Lake Superior synclinorium. The close folding such as exists in this district would thus be naturally accounted for as well as the very gentle folding along the Mesabi range which is supposed to be on the north limb of the synclinorium. The south limb, which in Michigan and Wisconsin is formed by the rocks of the Penokee-Gogebic district, is as yet unknown in Minnesota. The extensive areas of granitic rock in the St. Cloud and Mille Lacs Lake regions may be south of the synclinorium or, as believed by Zapffe,[4] they may represent intrusions within it. It is known that some of these granites do intrude metamorphosed sediments, but the age of the latter is not definitely established. If the granite is lower-middle Huronian or Laurentian in age and represents the basement upon which the Cuyuna metamorphosed sediments rest, then the south limb of the synclinorium may be expected to run northeastward from Little Falls through the northern part of Mille Lacs Lake toward Kettle River. On the other hand, if the granite is of later age than the Cuyuna series the south limb of the synclinorium may be far to the southeast.

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[4] Carl Zapffe: *op. cit.*

## Iron- and Manganese-bearing Rocks and Ores

### *Distribution*

As has been mentioned, seven or eight main belts of iron- and manganese-bearing formation run in an approximately northeast direction through the district. Those which have so far been found to contain orebodies of commercial importance are largely confined to the area south and east of the Mississippi River. However, some orebodies have been found in the iron-bearing areas in Morrison County west of the Mississippi River and in those in Crow Wing County north of the Mississippi River. As yet no attempts have been made to develop them. The northern of the iron- and manganese-bearing belts south of the river are included in the north range and the southern belts in the south range.

The most northerly of the north range iron- and manganese-bearing belts is only indefinitely known. Traces of iron-bearing formation occur in section 25, T.47 N., R.30 W., and thence northeastward in sections 20 and 22, T.47 N., R.29 W. beyond which it crosses the Mississippi River. No important orebodies have been found along it.

The next iron- and manganese-bearing belt to the south is of considerable importance, containing the manganiferous iron-ore bodies of the Ferro and Algoma (formerly the Hoch) mines in sections 32 and 33, T.47 N., R.29 W. and other important orebodies in section 28 near Manganese and in sections 22, 27, and 23, T.47 N., R.29 W. in the vicinity of Island Lake. Certain breaks occur in the continuity of the belt, but in general it is fairly well known because of extensive exploration. The iron-bearing formation appears to be more or less manganiferous throughout. It is bounded on the north mainly by green slate and on the south by graphitic slate.

South of this belt of manganiferous iron-bearing formation and ore there are a number of scattered occurrences of iron- and manganese-bearing formation in the region north and west of Menomin Lake and southwest of Rabbit Lake, such as those in sections 34 and 35, T.47 N., R.29 W. The relation between the separate areas is not yet known. To the east of them, south of Rabbit Lake, is the important iron-bearing belt containing the Kennedy orebodies. This belt is known from the southern part of section 25, T.47 N., R.29 W. as far east as the eastern part of Rabbit Lake where it either dies out or turns to the south and doubles back on itself.

South of the eastern end of the Kennedy iron-bearing belt is the northeastern end of another iron-bearing belt which lies to the south. Whether these two belts are actually connected by a synclinal turn is not yet possible to say. The southern belt runs southwestward through the northwestern part of section 32 and the centers of section 31, T.47 N., R.28 W.

and section 1, T.46 N., R.29 W. into the broad belt of iron- and manganese-bearing formation lying north and northwest of Ironton and Crosby in sections 2, 11, and 10, T.46 N., R.29 W. which contains the Mahnomen and Mangan No. 2 orebodies along the northern margin, and the northern Thompson, Armour No. 1 and Pennington orebodies along the southern margin. West of the Mahnomen and Pennington mines, the belt divides into two parts separated by a strip of slate and schist. The northern part continues a short distance southwestward into section 9, while the southern part extends southwestward along the north side of Blackhoof Lake in the southern part of section 9, where it contains the Hillcrest and other orebodies, and thence through the northwestern corner of section 16 and the northern part of section 17 into section 18, T.46 N., R.29 W. where the Rowe mine is located. Here it makes a synclinal turn northward and doubles back on itself for a short distance. The extent of the backward turn is not yet known.

This iron- and manganese-bearing belt is one of the most important in the district. It is known continuously over a length of more than 8 miles and along it are some of the most important ore deposits in the district. Unlike the belts to the north, this belt contains many important iron-ore deposits, though manganiferous iron ores also are found in it at various places. On the north, it is separated along the central portion from the Cuyuna-Mille Lacs-Sultana manganiferous iron-bearing area by a thin band of slate or schist, while on the south another thin band of schist separates it from the important Croft-Armour No. 2 iron-bearing belt.

The Cuyuna-Mille Lacs-Sultana manganiferous iron-bearing area is in section 3, south and southeast of Menomin Lake. It is manganiferous throughout and contains important bodies of manganiferous iron ore. It is known to be fairly wide near the Sultana and Cuyuna-Mille Lacs mines, but its longitudinal extent northeast and southwest is only imperfectly known. Areas of iron-bearing formation, however, occur north and south of June Lake in the northern part of section 9 and these may have some connection with the Cuyuna-Mille Lacs-Sultana area. Other orebodies occurring in the Cuyuna-Mille Lacs-Sultana ore-bearing area are those of the Mangan No. 1 and the Hopkins mines, the latter containing both iron ore and manganiferous iron ore.

The Croft-Armour No. 2 belt has been well explored to the northeast as far as section 1, T.46 N., R.29 W., where the Croft mine occurs. From this point it runs southwestward through the Meacham, Thompson, Armour No. 2, and Ironton properties in section 11, T.46 N., R.29 W., and continues a short distance into the eastern part of section 10, where it pinches out. The belt is fairly narrow throughout its length. North of the Armour No. 2 and southern Thompson orebodies which occupy its central portion in section 11, is a band of green schist which separates it from the Pennington-Armour No. 1 iron-bearing area.

The Croft-Armour No. 2 belt consists very largely of iron-bearing formation and associated iron ore, though locally, as in the eastern part of the Armour No. 2 property and the northern part of the southern Thompson orebody, manganiferous iron-bearing formation and ore occur in it. The proportion of ore to iron-bearing rock in the belt is high, the ore being largely a somewhat siliceous, hydrated, brown to red hematite, but in places containing important masses of red, medium soft, high-grade hematite, some of which is of Bessemer quality. The ore-bearing belt is bounded on the south mainly by green chloritic schist and red hematitic schist.

South of the Croft-Armour No. 2 and the Pennington-Rowe iron-bearing belts are local occurrences of manganiferous and non-manganiferous iron-bearing formation and ore, such as those in sections 19, 17, 12, and 1, T.46 N., R.29 W. These are the southernmost areas of iron-bearing formation in the north range.

The south range iron-bearing belts, although narrow, are far more extensive longitudinally than the iron-bearing belts of the north range. Of the known occurrences of south range iron-bearing formation those farthest northeast are in the Rice River region northeast of Aitkin. From this point southwestward as far as Deerwood many areas of iron-bearing formation are known. They are approximately along the same line of strike, but large unexplored areas are found between them. From Deerwood southwestward the south range iron-bearing formation has been traced more or less continuously to a point a short distance southwest of Barrows, a distance of about 24 miles. Mines and exploratory shafts are found along it at intervals, but many orebodies of importance remain undeveloped.

Although throughout most of this extent, the south range iron-bearing formation occurs as a single band, there are locally small stretches where two or three parallel bands occur. There are also numerous places along the belt where the iron-bearing formation has not yet been found and barren stretches occur which apparently break the continuity of the iron-bearing belt. To what extent these breaks indicate absence of iron-bearing formation and to what extent they indicate incomplete exploration has not been determined. South and southeast of Brainerd and south of Barrows a second iron-bearing belt has been shown to exist, running parallel to and about 1½ miles south of the main south range belt. Orebodies have been found locally along it.

The Morrison County belts of iron-bearing formation mentioned above are the continuations of the south range belts west of Mississippi River. Minor occurrences of iron-bearing formation are known in the vicinity of Bay Lake and Clear Lake south of the south range.

The areas between the different belts of iron-bearing formation have up to the present been only very imperfectly explored. However, in

almost every place where drilling has been done, some variety of schist or slate has been encountered, such as chloritic, sericitic, quartzose, or graphitic schist or slate. Green and gray micaceous and chloritic schist, green slate and black graphitic slate appear to be by far the most abundant. Locally quartzite has been encountered but the amount is usually small, and apparently it forms local lenses. Elsewhere considerable areas of different kinds of basic or semi-basic igneous rocks have been found.

The orebodies are found at intervals along the iron- and manganese-bearing formation belts. In some belts they are very abundant, whereas in other belts they are almost absent. Even along single ore-bearing belts they are grouped more or less in certain parts with stretches of barren iron- or manganese-bearing rock between. Thus along the Croft-Armour No. 2 belt, ore is almost continuous throughout, while in Armour No. 1-Pennington-Rowe belt important orebodies are grouped in certain localities with stretches of ferruginous chert and ferruginous slate between. Other belts, as the main south range belt, have orebodies scattered along them at irregular intervals separated by barren areas consisting of ferruginous chert, ferruginous slate, amphibole-magnetite rock, or magnetitic slate.

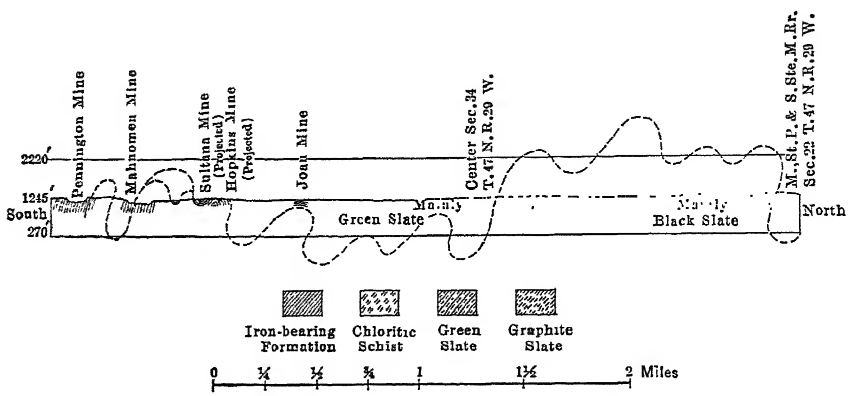


FIG. 2.—GEOLOGIC SECTION ACROSS THE NORTH RANGE, CUYUNA DISTRICT, SHOWING THE POSSIBLE STRUCTURE OF THE ROCKS.

The relation of the different iron-bearing and manganiferous iron-bearing belts to each other has not yet been determined. Several belts are known which represent the outcrops of opposite limbs of folds and it is believed that eventually all the different north range belts and possibly also the south range belts will be found to be thus related. Locally, it has been found that rocks between certain ore-bearing belts have characteristics which distinguish them from rocks between other ore-bearing belts. Thus in some areas graphitic slates predominate and elsewhere green slates or quartzose sericitic schists or phyllites predominate. Thus the relation between different ore-bearing belts may be determined by a

study of the associated rocks. Fig. 2 shows a north-south section through the north range with dotted lines to indicate the possible anticlinal and synclinal structures. The dotted lines should be regarded simply as a suggestion of what may be found.

### *Lithology and Occurrence*

The iron- and manganese-bearing formation of the Cuyuna district presents a variety of lithological types. Among the more common rocks composing the layers are hematitic and limonitic chert, hematitic and limonitic slate, cherty and slaty ferrous carbonate rock, green laminated slate, siliceous magnetitic slate, quartz-amphibole-magnetite rock, jaspilite, dark blue, red, brown, black and yellow iron ore, black, red, or brown manganiferous iron ore, green chloritic schist, and dark red hematitic schist.

Limonitic and hematitic chert are very abundant in the north range where they compose the principal part of the iron-bearing layers. They usually consist of white, pink, or gray chert interlaminated with hematite or limonite. The hematite or limonite laminæ may be pure iron oxide or they may be very siliceous. Frequently the iron oxide present in the laminæ is rather of the nature of a stain or an impregnation. Thus there are all gradations of ferruginous cherts from siliceous iron ore to chert containing only a small percentage of iron oxide. Frequently instead of being banded the ferruginous cherts present a blotched appearance due to the irregular distribution of chert and iron oxide or to the irregular staining of the chert by the latter. The more siliceous iron-stained chert usually shows a more irregular distribution of chert and iron oxide.

The chert laminæ in ferruginous chert are typically fine-grained, dense and flinty, but locally they have suffered partial disintegration and present a fine sugary appearance. When completely disintegrated they break up into fine sand, giving rise to what is known as wash ore. Wash ore is soft ferruginous chert from which the disintegrated chert can be removed by washing while the iron oxide remains behind and becomes concentrated.

The ferruginous chert occurring in the Cuyuna district is very similar to that which is found in most of the other Lake Superior iron-ore districts except the Mesabi. In all the Michigan and Wisconsin districts, the common rock composing the iron-bearing formation is a more or less regularly banded ferruginous chert which with the associated ferruginous slate is believed to have been derived by alteration from original cherty and slaty iron carbonate rocks. In the Mesabi district, however, the ferruginous chert is largely of the type known as "taconite," a rather massive, bedded cherty rock irregularly speckled with iron oxide which is supposed to have resulted from the oxidation of the ferrous silicate

rock "greenalite." No taconite or greenalite have been found in the Cuyuna district, although certain jaspilite layers have oolitic textures like taconite. It is, therefore, supposed that here, as in most of the other districts, the original rock from which the present hematitic and limonitic chert and slate and iron ore have in large part been formed is a banded cherty and slaty ferrous carbonate rock.

Hematitic and limonitic slate are not as abundant in the district as ferruginous chert, but occur locally as beds and lenses interlayered with ferruginous chert, iron ore and manganiferous iron ore. They are usually reddish, brownish, or yellowish in color and are finely laminated. Siliceous phases of the slates occur also which show gradation into ferruginous chert. These are thinly laminated but are hard and frequently contain thin chert layers. The ferruginous slates represent stages in the deposition of the iron-bearing formation when fine, argillaceous sediment rather than silica was being deposited with the iron. They are not as abundant in the Cuyuna district as in some of the other iron-ore districts, especially the Mesabi.

The ferruginous slates differ from the hematitic schist to be described later, in that they are of sedimentary origin and have been deposited with and suffered the same alterations as the ferruginous cherts. The hematitic schist, on the other hand, results from the alteration and impregnation with iron oxide of the green chloritic schist, a rock probably of igneous origin, which is commonly associated with the iron-bearing formation layers.

Cherty and slaty iron carbonate rocks are believed to be the original rocks from which the iron-bearing formation of the Cuyuna Range in its various phases is largely derived. Such rocks have been reported from many parts of the district in the deep drilling. Several different types occur with gradational phases between them. They vary in composition and texture and range from light to dark green or gray in color. Usually they consist of interlayered chert or argillaceous material with amphibole and iron carbonate. Magnetite is a common constituent and with increasing magnetite and recrystallization of the other constituents these rocks grade into amphibole-magnetite rock and magnetitic slate.

Cherty and slaty iron-carbonate rock generally shows distinct banding and many layers show fine lamination, the latter being characteristic especially of the slaty phases. The banding is largely due to interlayering of different constituents. The laminæ of chert and siderite are usually light gray or greenish in color, while those of argillaceous material are darker and with increasing magnetite become almost black. Some phases of iron-carbonate rock are light gray in color and consist almost entirely of chert and siderite which may be intermixed, or they may be more or less segregated into banded layers. Grains of magnetite often occur along lamination planes, making the banding more marked.

Other phases of iron-carbonate rock contain considerable amphibole and these show interbanding of light gray and somewhat darker green layers. Still other phases are very dark gray or green and consist mainly of amphibole and argillaceous material mixed with more or less siderite but containing little or no chert. Such phases are usually very fine-grained and show no marked banding. All these various types of iron-carbonate rock occur interbedded with each other and grade into each other.

With increasing argillaceous material, slaty iron carbonate rock grades into green laminated slate. This is a light to dark green rock generally showing very perfect and regular lamination of light and dark layers and consisting principally of finely crystalline amphibole. Most of it has slaty cleavage which lies at varying angles with the lamination planes. Green laminated slate is interbedded with cherty and slaty iron-carbonate rocks. Upon oxidation and impregnation with iron it gives rise to dark red ferruginous slate.

The siliceous magnetitic slate and amphibole-magnetite rock are phases of the original iron-bearing formation which have suffered metamorphic alteration and recrystallization. They are banded or laminated rocks, usually very dark-colored with a tinge of green. They consist mainly of amphibole, magnetite, and chert or quartz. The fine-grained, finely laminated types with slaty structures are known as magnetitic slates, while the more coarse-grained and coarsely-layered types are known as amphibole-magnetite rock. The banding is due to the segregation of the different minerals into layers and also to a difference in the coarseness of crystallization. Thus layers of fairly pure magnetite alternate with layers of amphibole, chert, or quartz, or mixtures of these minerals. Some bands are very fine-grained, especially those consisting mainly of magnetite, while other bands, as those of amphibole, are often medium coarse-grained. In some phases the silica is medium coarsely crystalline (quartz), and in other phases it is microcrystalline (chert). The segregation of minerals in the amphibole-magnetite rocks and magnetitic slates is never perfect. Small amounts of fine disseminated magnetite usually occur scattered through the quartz, chert, and amphibole layers, while quartz, chert and amphibole occur intermixed with magnetite along the borders of magnetite bands.

The composition of these rocks shows a wide range. In places they consist largely of magnetite and amphibole and elsewhere they contain a considerable amount of silica with amphibole, but with little magnetite. Carbon in the form of graphite is locally present. Irregularities in the texture are shown by the thickening or pinching of laminæ. These irregularities, however, are much less marked than they are in the ferruginous cherts.

The amphibole-magnetite rocks and magnetitic slate of the Cuyuna district differ from the typical amphibole-magnetite rock of other Lake

Superior iron-ore districts, such as the Mesabi, Gogebic, and Marquette, in that they are fine-grained, more perfectly laminated, and that in general they contain less quartz and more ferromagnesian minerals. The typical amphibole-magnetite rock of the Cuyuna district is a finely banded rock consisting of alternating bands of magnetite and amphibole with more or less chert or quartz. That of the eastern part of the Mesabi district, on the other hand, is an irregularly banded rock consisting of alternating layers of fine-grained magnetite and coarse-grained quartz and amphibole. In the Cuyuna district, however, highly quartzose and cherty phases of the amphibole-magnetite rock and magnetitic slate are also common.

Jaspilite is of rather rare occurrence in the Cuyuna district, but well-defined beds of it are found in several of the mines. Thus at the Cuyuna-Mille Lacs mine both the hard, red-banded jasper and the specular, schistose jasper occur along the south wall of the orebody where they are interlayered between typical banded hematitic chert on one side and low-grade manganiferous iron ore on the other side. The Hopkins mine also contains hard dense jasper.

Jaspilite is metamorphosed and recrystallized ferruginous chert. Where the metamorphism has not been very intense, a hard, dense, bright red rock is formed. This is the more common phase, while the crystalline, specular form which results from pronounced recrystallization is more local in its occurrence. Some jaspilite layers show a well-defined oolitic texture. In places in the district, very hard, dense, siliceous ferruginous chert occurs to which the term jasper is sometimes applied. This rock, however, varies in color from brown or reddish-brown to black and is simply a very dense form of the ferruginous chert.

The green and gray chloritic schist and the dark red hematitic schist do not belong to the iron-bearing formation proper. They usually occur as small lenses within the iron-bearing formation layers or they may bound the iron-bearing layers on either side. The dark red hematitic schist usually occurs along the borders of such lenses where they are in contact with the iron-bearing rocks or iron ore and have become impregnated with hematite for a variable distance from the contact while the green or gray chloritic schist occurs in the center of such lenses or forms entire layers where they are not in contact with iron-bearing formation.

As has been mentioned, it is believed that the chloritic schists represent original masses of basic igneous rock which were intruded as sills into the iron-bearing formation and associated sedimentary rocks while these were still largely in an unmetamorphosed condition. Some of them may have originated as local surface flows. Subsequent metamorphism and deformation were suffered alike by igneous rocks and sediments.

The Cuyuna orebodies are as a rule roughly tabular in shape, the longer axes being parallel to the bedding of the inclosing rocks. As the

beds usually dip steeply, the orebodies are shown at the surface as bands which extend for considerable distances along the strike of the beds. They vary in width up to several hundred feet and are usually continuous, some of the known orebodies being more than a mile long. Their behavior with depth varies. Some are comparatively shallow, extending perhaps to a depth of 100 or 200 ft. (30 to 60 m.) below the base of the glacial drift, where they give place to ferruginous chert, amphibole-magnetite rock, or unaltered iron-bearing formation. Others extend to greater depths, either continuing directly downward along the bedding or running diagonally downward parallel to the strike in the form of shoots. The data regarding the depths reached by the orebodies is still very incomplete. The deepest mining operations at present extend about 250 ft. (45 m.) below the base of the glacial drift, and to this depth ore is known to continue uninterruptedly. The drilling in the district has for the most part been shallow, few holes occurring which reach a depth greater than 700 ft. Ore, however, has been encountered in some of the deeper holes at depths up to 650 ft. below the base of the glacial drift and it is supposed that certain orebodies go much deeper.

The orebodies are usually inclosed between walls of ferruginous chert, ferruginous slate, or other iron-bearing rocks. Where ferruginous chert forms the wall rock, the contact is usually very irregular. Many beds and irregular horses of ferruginous slate and chert occur in the orebodies. Green chloritic schist, green laminated slate, and dark red hematitic schist and slate are also common associates of the iron ore and manganiferous iron ore. They may form the wall rock of the orebodies or they may occur interlayered with the ore, more or less parallel to the bedding. Other rocks associated with the orebodies are amphibole-magnetite rock, magnetitic slate, ferrous carbonate rock, and black carbonaceous slate. These are especially common along the south range where they form the wall rocks in many places. In the north range also, they have been found in many places underneath and along the walls of orebodies representing the unaltered portions of the iron-bearing formation.

Most of the iron ore and manganiferous iron ore occurring in the orebodies of the Cuyuna district is soft, but hard ore is also abundant both in the north and south ranges, and is more or less irregularly associated with the soft ore. The Cuyuna ore probably shows a greater variety of texture, composition, and color than the ore from any of the other Lake Superior iron-ore districts. It is associated with many different phases of the iron-bearing formation and varies in character and appearance according to the kind of iron-bearing rock with which it is associated. The iron ore shows all stages of hydration from pure reddish-blue hematite to ocherous, yellow limonite and both argillaceous and siliceous phases are common. On the north range it varies from dark reddish-blue high-grade hematite to reddish and yellowish-brown siliceous or argillaceous

hematite and limonite, while siliceous and argillaceous manganiferous iron ore is of common occurrence Some of the rich hematite ore is finely crystalline, but most of the ore, especially the more hydrated material, is amorphous. Locally, wash ore is associated with the other ores. On the south range the typical ores are reddish-brown hydrous hematite, dark brown to black limonite, and yellow, ocherous limonite.

The manganiferous iron ores consist of a mixture of various manganese oxides associated with more or less hydrated iron oxide. In most manganiferous iron-ore bodies local masses of pure black manganese oxide are found, some of it crystalline, but this ore is usually present in minor quantity and occurs irregularly intermixed with manganiferous iron ore. The manganiferous iron ores generally vary in color from dark red or brown to black and are usually fine-grained or amorphous.

### *Geologic Relations and Origin of Manganiferous Iron Ores*

*Occurrence.*—The manganiferous iron-ore deposits of the Cuyuna district are confined largely to the north range, although at two or three places along the south range iron ore carrying up to 3 or 4 per cent. of manganese has been found. Recently manganiferous iron ores are reported to have been found also in the region between Aitkin and Emily north of the Mississippi River. No manganiferous iron ore has been mined outside of the north range, however.

The principal deposits of manganiferous iron ore which have been found in the north range are those occurring along the northern iron-bearing belts south, east, and north of Menomin Lake and west of Rabbit Lake. Scattered deposits, however, have been found also along the southern iron-bearing belt extending from the Rowe mine northeastward to the Kennedy mine and locally in the area south of this belt. Six manganiferous iron-ore mines shipped ore in 1916 or previous years, these being, from north to south, the Algoma mine (NE¼ of NW¼ sec. 33, T.47 N., R.29 W.), the Ferro mine (SE¼ of NE¼ sec. 32, T.47 N., R.29 W.), the Cuyuna-Mille Lacs mine (SE¼ of SW¼ sec. 3, T.46 N., R.29 W.), the Mangan No. 1 mine (NE¼ of SW¼ sec. 3, T.46 N., R.29 W.), the Sultana mine (SW¼ of SE¼ sec. 3, T.46 N., R.29 W.), and the Mangan No. 2 mine (NE¼ of NE¼ sec. 10, T.46 N., R.29 W.). In addition, manganiferous iron ore has been shipped in appreciable amounts from the following iron-ore mines: Mahnomen mine (NW¼ of NE¼ sec. 10, T.46 N., R.29 W.), Thompson mine (W½ of NE¼ sec. 11, T.46 N., R.29 W.), Armour No. 2 mine (S½ of NW¼ sec. 11, T.46 N., R.29 W.) and Hillcrest mine (NE¼ of SE¼ sec. 9, T.46 N., R.29 W.). Thus out of a total of 21 mines which have produced ore in the Cuyuna district 10 have yielded manganiferous iron ore. At the present time, four or five new manganiferous iron-ore properties are being developed and

ore will probably be shipped from these in the near future. Among them are the Merritt mine (SW¼ of NW¼ sec. 33, T.47 N., R.29 W.), the MacKenzie mine (SE¼ of SE¼ sec. 28, T.47 N., R.29 W.), and the Joan mine (SW¼ of NE¼ sec. 3, T.46 N., R.29 W.).

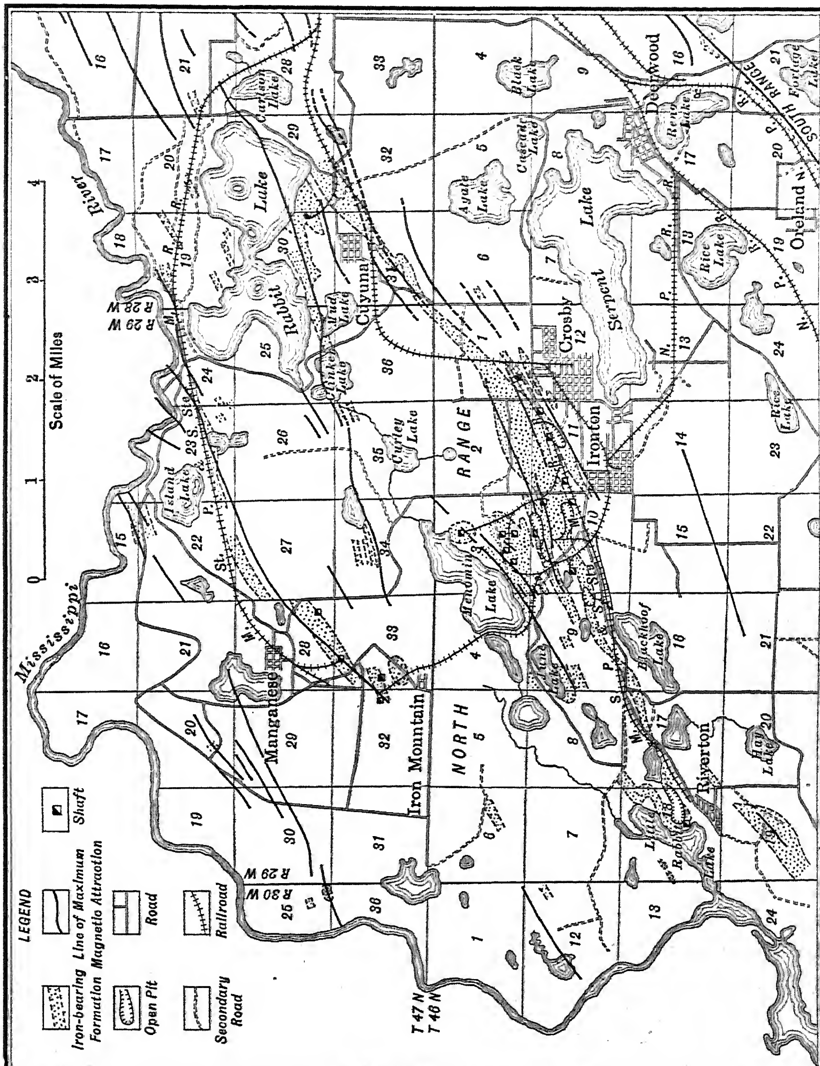


FIG. 3.—MAP SHOWING THE AREAS OF KNOWN IRON-BEARING FORMATION AND LINES OF MAXIMUM MAGNETIC ATTRACTION IN THE NORTH RANGE, CUYUNA DISTRICT, MINNESOTA.

In general occurrence, the manganiferous iron-ore deposits are similar to the iron-ore deposits, in fact there are many orebodies in the district in which both classes of ore are found. Like the iron-ore bodies, the deposits of manganiferous iron ore are roughly tabular in shape, the longer

diameters generally being parallel to the bedding of the formations inclosing them. They are, however, considerably more irregular in outline. Most of the manganiferous iron-ore bodies are associated with steeply dipping rocks and therefore dip at steep angles. Several deposits, however, have been found which occur along the axes of folds and these have a greater horizontal extent and are more irregular than bodies which occur on the steeply dipping limbs of folds.

Steeply dipping bodies of manganiferous iron ore are found in the Ferro, Algoma, and Mahnomen mines. In all these mines the general strike of the orebodies is approximately northeast and the dip averages between 60° SE and vertical. Although they have the same general attitude, there is a considerable difference in the form of the deposits in the three mines, due to the character of the material in which the ore occurs. At the Ferro mine the ore, which is of higher grade than most of the ore in the district, occurs along a comparatively narrow zone in manganiferous iron-bearing formation which is prevailingly cherty. Within this zone the richer manganiferous ore is generally concentrated along the hanging-wall side. In the Algoma mine the ore-bearing rock is also manganiferous and ferruginous chert, but here the ore-bearing zone is much wider than that at the Ferro mine. Ore does not occur throughout the zone, however, but is confined to four or five layers or lenses separated from each other by lean ferruginous chert or manganiferous ferruginous chert. The ore in the different lenses varies in character. In one lens it is low in phosphorus but high in silica, in another it is high in manganese and also moderately high in phosphorus, while in a third it is high in iron and only moderately high in manganese. The Mahnomen orebody differs from those of the Ferro and Algoma in having ore scattered throughout a comparatively wide zone. The ore is found in red slate or in yellow to brown clayey material probably derived from red slate, and occurs in irregular nodular masses or as irregular replacements along bedding and fracture planes. It is found in greater or less amount throughout the ore-bearing zone, being richer in the clay bands than in the red slate bands. Thus in these three mines, while the attitude of the beds is similar, the nature of the orebodies is quite different.

In the Cuyuna-Mille Lacs and Sultana mines, the orebodies are found along the axes of folds. The north orebody at the Cuyuna-Mille Lacs mine occurs along the crest of a southwestward pitching anticline, while the south orebody at the Cuyuna-Mille Lacs and that at the Sultana mine are found in a southwestward pitching synclinal trough. In both of these mines the ore-bearing zones are considerably wider than those at the other mines mentioned, but the ore is somewhat more irregularly scattered through them and does not occur as distinctly along stratigraphic horizons. The ore of the Cuyuna-Mille Lacs north orebody, similarly to that at the Algoma and Ferro mines, is associated chiefly

with manganiferous ferruginous chert. The situation of the orebody along the axis of an anticline suggests that it may occupy a fracture zone, the ore having replaced the fractured rocks. However, in its downward extension the orebody divides into two limbs, one south of the axis of the fold and one north, thus suggesting that a certain stratigraphic horizon was more amenable to replacement than the rest of the formation. The Sultana orebody resembles that at the Mahnomen mine in that the ore is associated with red slate and yellow and brown clay. Although of considerable width, the greatest extent of the deposit is northeast-southwest parallel to the strike of the rocks. Along the south side the rock beds dip steeply northwest while along the north side they dip more gradually to the southeast. The ore is found irregularly distributed, occurring as replacements and veins in the slate and clay. The south orebody at the Cuyuna-Mille Lacs mine occurs southwestward, along the strike of the Sultana orebody, and is similar to the latter in structure. The associated rock, however, is in general more cherty than at the Sultana mine.

In other deposits of manganiferous iron ore in the district there is apparently no definite relation between the structure of the rocks and the form of the orebody. Thus in the Armour No. 2 mine the rock beds have a general steep dip to the southeast. The Armour No. 2 orebody, however, consists in its southwestern part of rich red and blue hematite, in the northeastern part of dark brown to black manganiferous iron ore. The line separating these two types of ore is fairly sharp and cuts diagonally across the strike of the rocks. In other orebodies, such as the Hillcrest, manganiferous iron ore is found as irregular masses surrounded by iron ore. In such cases it probably represents local replacements in iron-ore deposits.

The borders of the manganiferous iron-ore bodies are usually indefinite and the outlines are irregular. There are all gradations between ore and the manganiferous or non-manganiferous iron-bearing wall rock. In places this gradation extends over only a few inches or a few feet, but elsewhere irregular masses of lean rock are found scattered through the ore many feet from the outer edge of the orebody.

*Size.*—The Cuyuna manganiferous iron-ore bodies are smaller than the iron-ore bodies and usually do not extend to as great depths. Deposits are known which show ore practically continuously over a width of 300 ft. (91 m.) and which extend along the strike for 1500 ft. (457 m.) or more. Usually, however, they are much smaller. The greatest depth to which any of the deposits have been worked is 175 ft. (53 m.) below the base of the glacial drift. Manganiferous iron ore has been shown by drilling to extend to greater depths than this locally, but by far the larger number of deposits will probably be found to have their principal concentration above this level.

In some mines ore-bearing zones of considerable width are found in which, however, the ore itself occurs only as thin beds or lenses or in irregular scattered masses separated from each other by lean manganiferous iron-bearing rock. Such beds or masses of ore range from a few feet to 40 or 50 ft. in thickness and have varying lengths and depths. In other mines the ore-bearing zones contain ore scattered throughout in small or large masses in a matrix of manganese and iron-bearing rock or clay. In such cases practically the entire deposit is mined, yielding a low-grade ore. However, even in these deposits certain portions are generally found to be richer than others, some parts being so lean that it does not pay to mine them.

In tonnage the manganiferous iron-ore deposits range on an average between 200,000 and 500,000 tons. Bodies containing more than 500,000 tons are exceptional, although some are known which contain 1,000,000 tons or more. Many small unworkable deposits are found. The lower limit of workability is variable, depending on a number of factors, such as grade and character of ore, shape of orebody, presence or absence of included rock masses, depth of surface drift, amount of water, etc. Where conditions are favorable, orebodies containing less than 200,000 tons can be worked. Many small bodies of manganiferous iron ore occurring in iron-ore deposits are mined in connection with the iron ore, whereas if such bodies were surrounded by lean rock they would be unworkable. As compared with the iron-ore deposits of the district which range in size up to 10,000,000 tons and over, the manganiferous iron-ore bodies are small. However, on account of the greater value of the ore, many small bodies which could not be mined if they consisted of iron ore have become commercially available.

*Associated Rocks.*—The rocks found in association with manganiferous iron ores are mainly those characteristic of the iron-bearing formations, including ferruginous chert, ferruginous slate, cherty and slaty iron carbonate, amphibole-magnetite rock and iron ore. Green laminated slate and black graphitic slate are also commonly associated with manganiferous iron ore, but green chloritic schist, which is common near many of the iron-ore deposits, has not been found in association with manganiferous iron ore.

Graphitic slate and green slate generally form the country rock, the manganiferous iron-bearing formation being interlayered with them. In several localities in the northern part of the district graphitic slate has been found on one side of an ore-bearing layer and green slate on the other side. Within the manganiferous iron-bearing layers and composing the major part of them are the other rocks mentioned above. The manganiferous iron-ore bodies form but a small part of the ore-bearing layers. Probably the most common of the associated rocks is ferruginous chert, but ferruginous slate also occurs abundantly. Ferruginous chert and

ferruginous slate usually form the wall rocks of the orebodies, but in many places they are also irregularly intermixed with the ore or occur in the orebodies as horses. Iron ore locally borders manganiferous iron-ore deposits, but more commonly it is found within them as lenses or irregular masses.

As regards their occurrence, the manganiferous iron-ore deposits of the district can be roughly separated into two groups: (1) those which are associated with ferruginous chert, and (2) those which are associated with ferruginous slate. The deposits of each class possess certain characteristics which distinguish them from the other class. In deposits associated with ferruginous chert, the ore is usually found along definite horizons which were probably originally occupied by layers of more easily replaceable rock, while in those occurring in slate the ore is generally more or less scattered throughout a wide zone. The ore associated with ferruginous slate is commonly amorphous, that in ferruginous chert is in many places finely crystalline and locally shows pure manganese minerals such as pyrolusite, psilomelane, and manganite in geodes and veins. Manganite veins, however, are also common in ores in ferruginous slate. Another fairly general distinction between the two classes is that the ore occurring in association with ferruginous slate is higher in phosphorus and lower in silica than ore found in ferruginous chert.

The ferruginous chert and slate associated with manganiferous iron ore may carry varying amounts of manganese. Locally they contain less than 1 per cent. and from this there are all gradations to ore carrying 25 to 30 per cent. manganese. In iron content the rocks also show great variation ranging from 15 or 20 per cent. to 45 or 50 per cent. Besides the iron and manganese, ferruginous chert contains mainly silica, while ferruginous slate contains chiefly silica and alumina.

Cherty and slaty iron-carbonate rocks are encountered below the ore deposits at varying depths. Also, they are sometimes found bordering orebodies or occurring within them. The iron-carbonate rocks associated with the manganiferous iron ores show a considerable range in composition and texture but nearly all of them are conspicuously laminated. In places they are very cherty and consist of interlayered chert, iron and manganese carbonates and some silicate minerals while elsewhere they are predominantly slaty, containing layers of argillaceous material, carbonates, and silicates, and but little chert. Magnetite is in places an abundant constituent. Locally scattered crystals of arsenopyrite have been found.[5]

It is noteworthy that much of the ferrous carbonate rock occurring in association with manganiferous iron ore contains considerable man-

[5] E. Newton: School of Mines Experiment Station, University of Minnesota, personal communication.

ganese carbonate. In general the manganese carbonate is not visible to the naked eye, being indistinguishable from the other carbonate minerals present. Frequently, however, one obtains specimens in which certain layers have a distinct pink tinge. It seems probable that in most cases iron, manganese, and perhaps other constituents are present in the form of mixed carbonates such as manganosiderite or ankerite. But associated with these there are doubtless also siderite, rhodochrosite, calcite, and other carbonates. Manganiferous iron-carbonate rocks may contain as much as 8 or 10 per cent.[6] manganese, but usually it averages less than 3 or 4 per cent.

Amphibole-magnetite rock and magnetitic slate are in many places associated with cherty and slaty iron-carbonate rocks, in fact most of the latter contain a varying amount of magnetite and silicate minerals. This indicates that very severe metamorphism accompanied the folding of the rocks. Intrusive igneous rocks which might have effected this metamorphism have not been found in the vicinity of manganiferous iron-ore deposits.

*Character and Grade.*—The manganiferous iron ore presents a variety of different phases depending more or less upon the nature of the rock with which it is associated. Some ore is black, some is yellowish, reddish or brownish-black. Some of it consists of fairly pure manganese oxides coarsely and irregularly intermixed with hydrated ferric oxides, giving mottled black and yellow, black and brown, or black and red ore, the color depending upon the degree of hydration of the ferric oxides. In places the intermixing is so coarse that pieces of high-grade manganese ore of considerable size can be extracted from the mixture, leaving a low-grade manganiferous iron ore. The iron and manganese oxides are generally amorphous and may be soft and porous or hard and massive. Locally, however, veins and geodes of various crystalline oxides are found. Other and more common phases of the manganiferous iron ore consist of a mixture of iron and manganese oxides so intimate that it is impossible to separate them mechanically. Such ores are usually uniformly black, reddish-black, or brownish-black and amorphous, the separate oxides composing them being indistinguishable. Between these two kinds of ore there are all gradations, manganiferous iron ore low in manganese being intermixed in all proportions with manganiferous iron ore high in manganese.

In still other phases of the ore, generally of lower grade, manganese ore instead of being mixed with iron ore may be intermixed with cherty or argillaceous iron-bearing or manganiferous iron-bearing rock. Thus there are manganese ores or manganiferous iron ores mixed with chert, slate, or clay which may carry varying amounts of manganese and iron.

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[6] E. Newton: personal communication.

Such ores are generally high in silica or alumina and with increasing amounts of these substances pass into manganiferous iron-bearing rock.

Both hard and soft ore occur in the deposits. The hard or soft character is not indicative of difference in grade. Some high-grade ores are hard, others are soft. Masses of almost pure manganese oxide may be quite soft or fairly hard, depending on the nature of the minerals composing them. Low-grade manganiferous iron ores also vary in hardness. It may be said in general that most of the hard ores are associated with cherty manganese-bearing and iron-bearing rocks and most of the soft ores with slaty or clayey manganese- and iron-bearing rocks.

The manganese minerals found in the Cuyuna manganiferous iron ores are manganite, psilomelane, wad, and pyrolusite, while the iron minerals are hematite and hydrated ferric oxides of the goethite-limonite series. As a general rule these various minerals cannot be distinguished in the ores, but locally where the separate minerals are distinguishable all of them have been found. In some mines, as the Algoma, veins and geodes of pure crystalline pyrolusite occur. Elsewhere veins of crystalline manganite or geodes consisting of successive layers of crystalline limonite or goethite with psilomelane and manganite, or psilomelane and pyrolusite are found. The order of formation commonly is: limonite-goethite, psilomelane, manganite, and pyrolusite. Where manganite is present pyrolusite usually is not found, the pyrolusite probably being an alteration product of manganite.

Besides the oxides there are present in manganiferous iron-ore deposits varying quantities of other minerals, both ore and gangue. Among the ore minerals are the carbonates, rhodochrosite and siderite or the mixed carbonates of iron, manganese, calcium, and magnesium such as ankerite, or of iron and manganese such as manganosiderite. These minerals are often present in appreciable amounts and in some mines form a considerable percentage of the ore. They are usually pinkish or yellowish-brown, finely crystalline, and are intermixed with more or less iron and manganese oxide. In most places the carbonates are probably partly altered phases of the original cherty or slaty manganiferous iron-carbonate rock, but locally they are formed by later impregnation, as veins of these minerals cut the oxidized iron-bearing formation.

The principal gangue material associated with the manganiferous iron ore is country rock in which the ore is concentrated. This is mainly ferruginous and manganiferous chert, red slate, and yellow clay. In depth unaltered phases of the iron and manganese-bearing rock, such as cherty and slaty carbonate and green slate, become common. Besides the rock impurities, however, certain gangue minerals form veins and local deposits in the ore. The principal one of these minerals is quartz, which is commonly found in veins alone or associated with manganite, psilomelane, ferric hydroxides or manganese and iron carbonates. In

some mines quartz is so abundant as to materially reduce the grade of the ore. Other gangue minerals found are calcite, iron silicates and in a few places barite. In one or two of the mines there are masses of fine-grained epidote the origin of which is uncertain. It is possible that among the gangue minerals there are also manganese silicates such as rhodonite associated with the iron silicates or with rhodochrosite. Such silicates, however, have not been definitely recognized.

As may be judged from the description of the varying character of the ore, there is also a great variation in the grade. This is true not only of different deposits but within short distances in the same deposit, especially across the strike of the formation. In large part these variations are due to original differences in the character of the beds in which the manganese and iron ore are now concentrated. There are also variations along the strike of the beds, however, which are not attributable to original differences of material, but rather to irregularities in ore concentration. Thus lenses of manganiferous iron ore may along the strike run into high-grade iron ore or irregular masses of iron ore may be inclosed within the manganiferous iron-ore deposits. In the same manner, masses of lean or barren rock may be inclosed with the orebodies. The result of this is that the manganese content of the orebodies may vary from 2 or 3 per cent. to 15 or 20 per cent. within very short distancee. The iron content varies similarly, but not so irregularly nor usually through so great a range in short distances. Silica and alumina are fairly constant in quantity along the same beds, but vary in amount in different beds according as the beds are cherty or argillaceous.

In view of these irregularities in the composition of the ore there is considerable difficulty in maintaining a uniform product from the mines. This difficulty is overcome, however, by mixing ores from different portions of the orebody and also by marketing several distinct grades of ore. Some of the mines offer as many as four different grades of manganiferous iron ore.

Iron and manganese oxides with silica and alumina form the principal constituents of the ore and in most cases their percentages, together with that of water of hydration, add up to approximately 100. In places lime and magnesia are present in considerable amount, but they are more or less local in their occurrence. In some ores, the amount of volatile material present is abnormally high, due in part to the high moisture content, and in part to the presence of carbonates. The iron content of the manganiferous iron ores usually ranges between 25 and 50 per cent. while the manganese content ranges up to 30 or 35 per cent. Silica is present up to 16 or 18 per cent. and alumina varies between 1.5 and 4 per cent. The combined water in the ore usually ranges between 8 and 12 per cent.; the higher-grade ores are lower in moisture. In deposits where the iron and manganese are present in large part as carbonates, carbon dioxide is found in considerable amounts.

In general it may be said that as regards grade there are two principal classes of manganiferous iron ores in the district: (1) low-phosphorus ores, usually high in silica, and (2) high-phosphorus ores which are generally moderately low in silica. Low-phosphorus ores are commonly

TABLE 1.—*Analyses of Manganiferous Iron Ores from the North Range, Cuyuna District, Showing Range in Percentage of Different Constituents*

| Mn | Fe | P | $SiO_2$ | Combined Fe and Mn | Mn | Fe | P | $SiO_2$ | Combined Fe and Mn |
|---|---|---|---|---|---|---|---|---|---|
| 1.01 | 50 75 | 0 276 | ..... | 51.76 | 18.55 | 39.78 | 0.056 | .. .. | 58.33 |
| 1.16 | 55.92 | 0.091 | 8 24 | 57.08 | 19.40 | 35.01 | 0 128 | .... | 54.41 |
| 1.49 | 50.66 | 0.122 | ..... | 52.15 | 19 76 | 35.48 | 0 041 | .. | 55 24 |
| 2.37 | 55.69 | 0.391 | ..... | 58.06 | 20.53 | 32 44 | 0.076 | 9 95 | 52.97 |
| 2.42 | 54.45 | 0.034 | .. | 56.87 | 20.73 | 31 25 | 0 075 | 12 94 | 51.98 |
| 2 44 | 56.42 | 0.134 | .... . | 58.86 | 21.70 | 33 52 | 0.027 | . .. | 55 22 |
| 3.13 | 58.76 | 0.123 | .... | 61.89 | 21.75 | 30 68 | 0.060 | .... | 52.43 |
| 3.61 | 51.36 | 0 058 | ..... | 54.97 | 21 80 | 30 27 | 0.211 | ... | 52 07 |
| 4.49 | 51.90 | 0.085 | 9 25 | 56.39 | 22 51 | 34 06 | 0.058 | 9.94 | 56 57 |
| 4.96 | 46.52 | 0 044 | ..... | 51.48 | 22 63 | 34 86 | 0.105 | ..... | 57.49 |
| 5.09 | 55.58 | 0 023 | . .. | 60.67 | 23 45 | 27 27 | 0 059 | .... | 50 72 |
| 5.16 | 50.64 | 0 107 | .. .. | 55 80 | 23 63 | 35 38 | 0.062 | 9 69 | 59 01 |
| 5.90 | 49.75 | 0.324 | .. . | 55.65 | 23 70 | 30 66 | 0 258 | ..... | 54.36 |
| 6.75 | 54 53 | 0 021 | ... . | 60 28 | 24.12 | 34 33 | 0.087 | ..... | 58.45 |
| 6.76 | 47.58 | ..... | 11 65 | 54.34 | 24.32 | 33 87 | 0.240 | ..... | 58.19 |
| 6.81 | 45.22 | 0.173 | ..... | 52.03 | 24 36 | 30.85 | 0.066 | ..... | 55.21 |
| 7.63 | 47.26 | 0.139 | ..... | 54.89 | 25.00 | 30 61 | 0.076 | ..... | 55.61 |
| 7.98 | 47.36 | ..... | 13.97 | 55.34 | 25.19 | 30.96 | 0.141 | ..... | 56.15 |
| 8.02 | 53.93 | 0.029 | ..... | 61.95 | 25 59 | 31.05 | 0.333 | ..... | 56.64 |
| 8.05 | 52.59 | 0.212 | ..... | 60.64 | 26.20 | 31 19 | 0.080 | ..... | 57.39 |
| 9.18 | 44.03 | 0.028 | ..... | 53.21 | 26.21 | 31.20 | 0.294 | ..... | 57.41 |
| 9.45 | 48.37 | 0.076 | .. .. | 57.82 | 26.62 | 36.55 | 0.051 | ..... | 63.17 |
| 9.48 | 47.03 | 0.325 | 3.18 | 56.51 | 27.15 | 31.91 | 0.064 | ..... | 59.06 |
| 10.72 | 44.07 | ..... | 12.39 | 54.79 | 27.52 | 28 27 | 0.156 | ..... | 55.79 |
| 10.84 | 39.38 | 0 061 | ..... | 50.22 | 27.88 | 23.20 | ..... | 16.22 | 51.08 |
| 11.08 | 44.41 | 0 021 | ..... | 55.49 | 28.11 | 27.94 | 0.172 | ..... | 56 05 |
| 11.51 | 45.90 | 0.152 | ..... | 57.41 | 28.75 | 31.56 | 0.083 | ..... | 60.31 |
| 12.14 | 41.07 | 0.138 | ..... | 53.21 | 29.51 | 22.03 | 0.121 | ..... | 51.54 |
| 12.31 | 41.96 | 0.036 | ..... | 54.27 | 29.92 | 22.71 | 0.073 | ..... | 52.63 |
| 13.91 | 38.05 | 0.050 | ..... | 51.96 | 30.10 | 28.89 | 0 041 | ..... | 58.99 |
| 13.95 | 40.90 | 0.023 | ..... | 54.85 | 31.46 | 28.22 | 0.043 | 6 72 | 59.68 |
| 14.65 | 40.56 | 0.045 | ..... | 55.21 | 31.50 | 32.42 | 0.063 | ..... | 63.92 |
| 14.90 | 38.48 | 0.279 | ..... | 53.38 | 32.40 | 31.24 | 0.038 | ..... | 63.64 |
| 14.91 | 38.20 | 0.307 | ..... | 53.11 | 33.28 | 25.48 | 0.034 | 8.85 | 58.76 |
| 15.06 | 38.79 | 0.046 | ..... | 53.85 | 33.44 | 18.08 | 0.088 | ..... | 51.52 |
| 15.37 | 34.74 | 0.062 | 15.10 | 50.11 | 33.66 | 27.36 | 0.129 | ..... | 61.02 |
| 16.30 | 35.00 | 0.140 | ..... | 51.30 | 34.10 | 22.34 | 0.046 | ..... | 56.44 |
| 16.96 | 38.67 | 0.088 | ..... | 55.63 | 34.83 | 24.69 | 0.059 | 9 29 | 59.52 |
| 17.18 | 33.71 | 0.041 | 13.44 | 50.89 | 35.85 | 24.96 | 0.043 | ..... | 60.81 |
| 17.55 | 39.25 | 0.255 | ..... | 56.80 | 36.90 | 17.72 | 0 032 | ..... | 54.62 |
| 18.00 | 32.37 | 0.198 | 13.60 | 50.37 | 37.94 | 19.93 | 0.071 | ..... | 57.87 |
| 18.00 | 37.03 | 0.085 | 12.72 | 55.03 | | | | | |

associated with cherty iron-bearing formation and high-phosphorus ores with slaty iron-bearing formation. Because of their association with different kinds of rocks, high-phosphorus orebodies and low-phosphorus orebodies generally occur separated from each other. This association, however, does not hold universally, and in some places high- and low-phosphorus ores occur together in the same series of rocks or even in the same deposit. Some mines produce only one class of ore, while in other mines orebodies of both classes are found closely associated.

In Table 1 are grouped a large number of analyses of manganiferous iron ores taken from practically all the known manganiferous iron-ore deposits in the north range. Many of them represent samples taken in drilling while many of them are of samples taken in mines. They were selected to show the general range in iron, manganese, and phosphorus content and in some cases the silica content as well. In most instances, in order to show the range of these constituents, several analyses are given of ores which have the same manganese content but which show differences in the percentage of iron and phosphorus. Only ores in which the combined iron and manganese content is more than 50 per cent. are included in the table, although much ore is being shipped at the present time in which the combined metallic content is considerably less.

The analyses show the great variation in the manganese content of the ores. They show also that ore containing a given percentage of manganese may vary greatly in the amount of the other constituents present, especially the phosphorus. Aside from minor irregularities, the iron content decreases with increase of manganese content, while the silica decreases with increase in content of combined metals, and *vice versa.* The phosphorus content shows no such regularity, the variations being independent of the iron and manganese content. In general, however, it may be said that the ores which are highest in manganese are usually of the class of low-phosphorus ores.

*Origin.*—The genesis of the manganiferous iron ore of the Cuyuna district is in general similar to that of the iron ore. There are, however, certain minor differences due to the different behavior of iron and manganese under oxidizing conditions. In order to bring out these differences more clearly, it may be well to outline briefly the principal steps in the formation of the iron ore.

1. The original iron-bearing solutions, the source of which may be disregarded in this discussion, deposited their iron under reducing conditions mainly as ferrous carbonate but perhaps in part as ferrous silicate. During the precipitation of the iron salts, colloidal silica and clay carried by the water were deposited also. Sometimes the iron salts were deposited in greater quantity and sometimes the clay or silica, thus giving rise to successive laminæ of these various materials. Fine laminæ of

silica or of clay became interlayered with laminæ of iron salts. The general character of the sediments varied according to the material predominating in the deposits. When clay was in excess a sediment was formed which ultimately gave rise to green slate or slaty iron carbonate, and when colloidal silica was in excess a sediment which developed into cherty iron carbonate was formed. However, these different types occur interlaminated or interbedded.

It seems probable that not all the iron was deposited in the ferrous form but that locally oxidizing conditions prevailed and iron was deposited as ferric hydroxide. As in case of the iron-carbonate deposition, clay and colloidal silica were probably present and were deposited in greater or less amounts. Upon consolidation, these sediments gave rise to ferruginous slate and ferruginous chert with perhaps some beds of fairly pure hematite or hydrated hematite. Such rocks would be similar in general to the rocks formed by the surface oxidation and weathering of cherty and slaty iron-carbonate rocks. It is believed, however, from the evidence afforded in various parts of the district, that the iron deposited in the ferric form originally was very small in amount.

2. Consolidation of the sediments took place and igneous rocks became associated with them mainly as local intrusive sheets and sills, but perhaps in part also as extrusives.

3. All the rocks were profoundly folded and metamorphosed. The argillaceous rocks and many of the igneous rocks were rendered schistose, producing green slate, chloritic schist, and other associated rocks. The iron-carbonate rock was recrystallized and much of it was altered to amphibole-magnetite rock and magnetitic slate. Where original beds of ferruginous chert were present they were altered to jaspilite. The layers of oolitic cherty ferric oxide in the jaspilite were probably original and were not formed by the alteration of ferrous silicate (greenalite) as is the case with similar material known as taconite found in the Mesabi district. This is indicated by the fact that apparently there was no intervening erosion period during which the alteration of the greenalite could have taken place before the metamorphism to jaspilite.

4. Later intrusions of igneous rocks occurred locally which probably caused further metamorphic changes.

5. The rocks were elevated and eroded, the upper portions being exposed to processes of weathering and oxidation. The cherty and slaty ferrous carbonate rocks were altered to hematitic and limonitic chert and hematitic and limonitic slate for varying distances from the surface. The green slate and chloritic schist associated with the iron-bearing rocks were altered to ferruginous slate and hematitic schist respectively. However, oxidation was not the only result of surface action. Meteoric waters percolating through the rocks dissolved and carried away silica, leaving the iron oxide more concentrated. In much of the ferruginous

chert in which the iron content was originally fairly high and which was favorably situated with reference to underground circulation, this process resulted in the formation of iron-ore deposits. Ferruginous slates, on the other hand, on account of their high alumina content and imperviousness to water, do not give rise to iron ores. The iron-ore deposits, however, were probably not formed exclusively by the removal of silica. There is considerable evidence to show that to some extent there has also taken place a movement of the iron. Ferric oxide is in many places found as veins cutting across beds of iron-bearing rock, as irregular impregnations in rocks which originally were free from ferric oxide or contained but

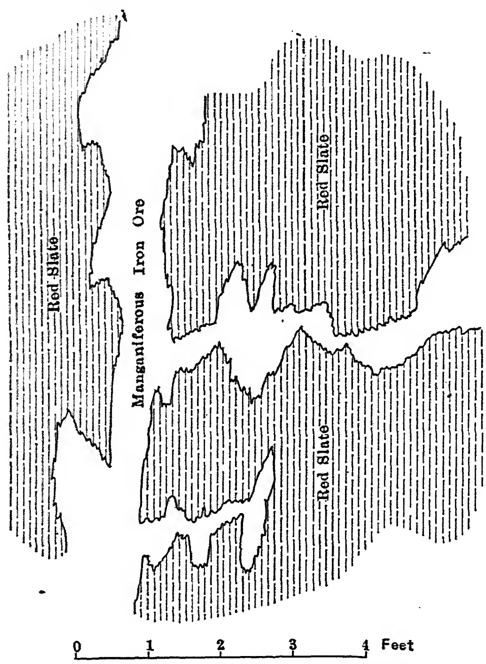


FIG. 4.—SKETCH SHOWING THE REPLACEMENT OF FERRUGINOUS SLATE BY MANGANESE OXIDE ALONG BEDDING PLANES AND ALONG FRACTURE PLANES AT RIGHT ANGLES TO THE BEDDING PLANES.

small amounts, and in the crystalline form in geodes. While all these phenomena show that movement of iron has taken place, there is little doubt that the principal concentration in the formation of iron-ore deposits resulted from the removal of silica.

In the preceding paragraphs are stated briefly the various steps which have led to the formation of the iron-ore deposits. The manganiferous iron ores have passed through practically the same stages in their formation as the iron ores. The original cherty and slaty iron-carbonate rocks which have been encountered locally in the manganiferous iron-bearing

beds in association with the ores, nearly always show small and varying percentages of manganese present in the form of rhodochrosite (manganese carbonate) associated with siderite or in the form of the double carbonate of iron and manganese, manganosiderite. These manganiferous iron-carbonate rocks have undergone the same changes that have been described for the iron-carbonate rocks. They have been dynamically metamorphosed, resulting in recrystallization and in the development locally of considerable magnetite and amphibole. Upon exposure to processes of weathering and oxidation they were altered to manganiferous ferruginous chert, manganiferous ferruginous slate, and manganiferous iron ore.

While the formation of the manganiferous iron-ore deposits has in general been similar to that of the iron-ore deposits, there are certain differences and these have resulted in differences in the nature of the two classes of deposits. The iron-ore deposits, as stated, were formed mainly by the removal of silica from ferruginous chert rich in iron. Some solution and redeposition of iron doubtless took place during its formation, but it was of minor importance. In the formation of manganiferous iron-ore deposits, on the other hand, the solution and redeposition of manganese has played a very important part, although in this case the removal of silica also has been a factor in the ore concentration. As a result of the predominance of solution and redeposition, the manganiferous iron-ore deposits have been formed not in ferruginous chert alone, but also in ferruginous slate. Many important deposits occur in ferruginous oxide along fracture planes, bedding planes, cleavage planes, and other water channels, or as irregular nodular impregnations throughout the mass of the rock. While iron-ore deposits as a rule form large bodies containing locally large and small masses of residual rock, the manganiferous iron-ore deposits are more largely rock, *i.e.*, ferruginous chert, slate and clay, in which the ore occurs as large and small irregular impregnations, nodules, veins, geodes, and replacements. The proportion of ore to rock varies greatly in manganiferous iron-ore deposits; locally ore masses are practically continuous with one another while elsewhere they are scattered. In general there is much more rock in manganiferous iron-ore deposits than in iron-ore deposits.

The principal evidence of the importance of solution and redeposition in the formation of manganiferous iron-ore deposits is: (1) occurrence of veins of manganese minerals and impregnation of the adjacent rock by manganese oxide for varying distances from the veins; (2) similar replacement of rock by manganese oxide on both sides of fracture planes, bedding planes, or other openings allowing the passage of water (Fig. 4); (3) the gradual decrease in manganese content of such replaced material away from the fissures; (4) the abundant occurrence of geodes and veins containing crystalline manganese oxide; (5) the occurrence of concentric

layers of different manganese minerals in such crystalline material; (6) the prevalence of nodular and concretionary forms in the ore deposits; and (7) the close association of the manganese oxides with vein quartz.

Just how important the removal of silica has been in the formation of manganiferous iron-ore deposits is uncertain. Doubtless its importance is considerable in those deposits which are found in association with manganiferous and ferruginous chert, but even in these, evidence of solution and redeposition of manganese exist. In the deposits occurring in ferruginous slate or clay, the leaching of silica has probably not been a very important factor in the concentration of the ore.

It appears, therefore, that the same processes have been active in the formation of both iron-ore and manganiferous iron-ore deposits in the Cuyuna district. The two metals, however, have been affected somewhat differently. On account of the greater solubility of manganese, its distribution through the rocks has been greater. It has travelled farther and is not so closely confined to the layers that contained it originally. However, the solubility is apparently only slightly in excess of that of iron, and for this reason the two metals in general tend to be concentrated together. As already stated, there is also a small amount of iron dissolved and reprecipitated. The differences noted, therefore, are differences of degree and not of kind.

## Discussion

The Chairman (E. G. Spilsbury, New York, N. Y.).—The relation of the character of manganese ore to the surrounding rocks was called to my attention very strikingly in a recent investigation in Costa Rica. In the foothills of the main coastal range there is a large deposit of manganese which extends for over 150 miles along the coast. This is in a porphyry district and the occurrences are almost exclusively along the lines of contact between a recent diorite intrusion through the porphyries. Wherever that intrusion appears to have exerted great force, the line of contact is very clear between the porphyries and the diorites. Where the action has been slower and the lifting force of the intrusion more gradual, there has been a zone of fracture of very considerable width.

Those ores which occur where the fractured zone has not been very extensive carry a lower percentage of manganese than where it has been deposited along a wider zone of fracture. In the so-called narrow veins, which are really the predominant ones, the solution carrying the manganese has evidently attacked the silica in the intrusive rock, and the resulting ore is practically a combination of manganese and silica, running from 20 to 30 per cent. of manganese and from 20 to 40 per cent. of silica: hence those ores are not likely for a long time to become a factor in our manganese supply for this country.

On the other hand, where the disturbance to the porphyritic schist has been more intense, the manganese deposits are not so extensive, but the ore is richer, running from 45 to 51 and 52 per cent. manganese, with a silica content of from 4 to 5 per cent. There seems to be some resemblance in these deposits to those of the Cuyuna range.

Edmund Newton, Minneapolis, Minn.—The State Bureau of Mines has been working on the problem of concentrating manganiferous iron ore for two years or more. I should consider hand sorting as the best form of treatment, because it is practised at all of the mines which ship ore for making alloys, but not, however, on the ore that is being used for pig iron. Hand sorting, assisted by selective mining, is readily applicable to some of this ore because the gangue is practically the original rock and is easily separated. However, it is only in rather limited zones that we find clean patches of rock. Generally the silica exists as a skeleton of the original rock structure, in which the manganese has been precipitated; some of the silica may have been dissolved, but if any of the original rock remains, the silica must be eliminated by ore-dressing processes.

Another problem is the separation of iron oxides from manganese oxides; this involves the same questions as the previous case, because most of the iron was disseminated through the original rocks, not in combination with silica, but as a mixture. Another difficulty is the removal of phosphorus from the high-phosphorus ores; this does not seem to be possible at the present time. We do not know just how the phosphorus occurs, although in a number of experiments recently we have noticed some crystals of apatite. Whenever we improve the percentage of manganese, there is always an increase in phosphorus.

We have even tried to extract the manganese by leaching. With 0.5 per cent. of sulphuric acid, we have been able to get, under proper conditions of temperature, possibly 75 per cent. of manganese. Our main difficulty is to precipitate it from the solution and recover it in some form that would be useful.

Chairman Spilsbury.—Have you ever noticed in these deposits whether all of the manganese is really pyrolusite or whether there are other combinations of manganese? The reason for my asking is that in the deposits in Costa Rica, where the fracturing has been more extensive, a great many of the cavities are filled with clay, and in that clay we find nodules of manganese which contain absolutely no silica, the silica having been replaced by lime; where the lime came from I have not been able to determine. The ore is extremely low in phosphorus, lower than the average run of the veins.

Edmund Newton.—In at least two of the Cuyuna deposits there is a type of material that is quite high in $CO_2$. In one of them there is a small quantity of peculiar material which shows distinct large crystal

facets and is rather glisteny and shiny. A complete analysis shows about 20 per cent. CaO, a corresponding amount of $CO_2$, and 7 to 10 per cent. of manganese.

In another, one of the best known deposits on the Cuyuna range, there appeared to be a distinct zone, possibly 10 or 12 ft. wide, which seemed to be the nucleus of the deposit; a good deal of the original rock is present and you can see that it contains manganese, although you would never judge it was manganese ore. We have been passing it by for a long while, not knowing that there was any high-carbonate material up there. Analysis showed that it contained 26 per cent. manganese, 23 per cent. $CO_2$, almost no iron, and only 22 per cent. silica. Whether this was some of the original deposition from the inland sea, originally very high in manganese, we do not know; we have been trying to find the continuation of that zone, but it seems to occur in one little corner only of the deposit.

# The Supposed Reversal of Inheritance of Ferrite Grain Size from that of Austenite

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(St Louis Meeting, October, 1917)

THE data which are collected in Table 1 show that the ferrite of low-carbon steel and of electrolytic iron, like the network of hypo- and hyper-eutectoid carbon steel, inherits, either absolutely or relatively, the grain size of the mother austenite from which it is born in cooling through the transformation range. This evidence is so abundant and concordant as to cast doubt on Prof. Jeffries'[1] hypothesis of what might be called reversed inheritance, by which he explains Mr. Ruder's observation[2] that a grain in a "slug of pressed pure electrolytic iron powder" was coarser, perhaps 1,500,000$\mu^2$, after a 3-hr. exposure to 1000° C. than after a 3-hr. exposure to 1250–1300°, when the grain size was about 35,000$\mu^2$. His explanation implies a natural tendency of coarser austenite grains to yield finer ferrite ones than spring from finer austenite grains, so that the inheritance is reversed.

What seems to me a simpler explanation is that the enormous grains which followed the 1000° heating were not inherited from the mother austenite, but were formed by the process of natal coarsening, by means of which Stead and Carpenter developed grains of this same order of magnitude in this same kind of material, on heating to 950° or to 1000°, the temperature used by Mr. Ruder, provided the exposure was not unduly prolonged, and it does not seem to have been in Mr. Ruder's case. That their coarsening was natal and not through inheritance is shown first by the fact that their grain coarseness was out of all proportion to their relatively low temperatures and short periods, and hence was very far in excess of the grain size which the mother austenite must have had, and second and more cogently by the fact that the shape of these coarsened grains was evidently determined by the temperature distribution which existed during and immediately after the $Ar_3$ transformation, that during the passage from the gamma to the non-gamma state.

In discussing this subject, we must remember that coarse ferrite grains, whether those formed in cooling from the molten or those induced by

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[1] *Trans.* (1917), **56**, 594, 597.

[2] *Op. cit.*, 591, 592.

Stead and Carpenter's process of natal coarsening, break up on transforming into austenite at $Ac_3$ on any later heating, the coarsening of ferrite not being inherited by the austenite. Further, that the size of the new ferrite grains formed in case natal coarsening does not occur, depends on the grain size not of the initial ferrite but of the mother austenite, and hence increases with the length and height of heating beyond $Ac_3$. Hence, in Mr. Ruder's second or 1300° heating, the coarse initial grains of ferrite formed natally in the $Ar_3$ transformation after his prior 1000° heating, and representing not that heating but that transformation, broke up on rising past $Ac_3$ in his next heating, which was to 1300°. The ferrite grains which would next form on cooling past $Ar_3$ would inherit the austenite grain size formed in this 1300° heating, and would have no tendency to resume the now effaced enormously greater size given natally during the prior $Ar_3$ transformation. I do not mean to assert that the austenite

TABLE 1.—*Selected Cases of Grain Growth*

| No | Date | Investigator | Carbon Content, Per Cent | Behavior of Specimen in Prior or Later Treatment | Behavior of Specimen in Present Treatment: Treatment: Exposed to °C. | Length of Exposure: Hr. | Length of Exposure: Min | | Average Grain Size in $\mu^2$ | Summary of Coarsening |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | GROUP I. HEREDITARY COARSENING A. Low-Carbon Steel | | | | | | |
| 1 | 1900 | Fay and Badlam | 0 07 | | 1,143 | .. | ... | | 600 | Slight to 1,125° then greater to 1,247° |
| 2 | 1900 | Fay and Badlam | . .. | ...... | 1,247 | .. | ... | .......... | 20,000 | |
| 3 | 1911 | Howe | 0 027 | .......... | 1,200 | 1 | ... | .......... | 6,900 | Slight at 1,200°, marked at 1,300°. |
| 4 | 1911 | Howe | .... | .......... | 1,300 | 1 | ... | .......... | 37,000 | |
| 5 | 1916 | Pomp.... | 0.08 | .......... | 1,100 | 8 | | | 1,480 | Slight and irregular up to 1,100°, great and nearly regular at 1,300°. |
| 6 | | | | | 1,200 | 8 | ... | .......... | 16,090 | |
| 7 | | | | | 1,300 | 8 | ... | .......... | 17,350 | |
| | | | | | B. Electrolytic Iron | | | | | |
| 8 | 1909 | Terry..... | ..... | .... ..... | 1,300 | ? | ... | .......... | 20,000 ± | Slight coarsening at 1,000°, far more at 1,200° and 1,300°. |
| 9 | 1911 | Howe..... | 0.023 | | 1,300 | .. | ... | .......... | 9,500 | |
| 10 | 1911 | Howe. ... | 0 029 | .... ..... | 1,300 | .. | ... | .......... | 9,800 | |
| 11 | 1914 | Storey Ingot B | .... | .......... | 950 | . | 30 | .......... | 7,400 | |
| 12 | | | .... | .......... | 1,050 | .. | 15 | .......... | 23,000 | |
| 13 | | | .... | .......... | 1,150 | 3 | ... | .......... | 22,900 | |
| 14 | 1917 | Ruder . | .... | Previously Coarsened at 1,000°, see No 29 | 1,250–1,300 | 3 | ... | .......... | 35,000 | |

TABLE 1.—*Selected Cases of Grain Growth (Continued)*

| No | Date | Investigator | Carbon Content Per Cent | Behavior of Specimen in Prior or Later Treatment | Behavior of Specimen in Present Treatment: Treatment: Expose to °C. | Length of Exposure: Hr. | Length of Exposure: Min. | | Average Grain Size in $\mu^2$ | Summary of Coarsening |
|---|---|---|---|---|---|---|---|---|---|---|
| GROUP II. THE NATAL COARSENING OF ELECTROLYTIC FERRITE | | | | | | | | | | |
| 15 | 1913 | | 0 008 | ........... | 950 | .... | 30 | Cooled to 885 then quenched. | Fine | Fine if quenched above $Ar_2$, coarse if quenched below. |
| 16 | 1913 | | 0 008 | ..... ... | 950 | .... | 30 | Cooled to 880 then quenched. | Coarse 4,400,000 | |
| 17 | 1913 | | 0.008 | ........... | 1,000 | .... | ..... | Quenched at 1000. | Fine | |
| 18 | 1913 | | 0.008 | .... ..... | 1,000 | .... | ..... | Quenched 2 or 3 sec. after cooling through $Ar_3$ | Coarse | |
| 19 | 1913 | Stead and Carpenter | 0 008 | ........... | 890 | 72 | .... | .......... | Fine | On slow cooling fine if $T$ max is below $Ac_3$, coarse if it is above |
| 20 | 1913 | | 0 008 | ..... .... | 910 | 1 | .. | .......... | 2,700,000*a* | |
| 21 | 1913 | | 0 008 | ........ . | 915 | 1 | .... | . ......... | 7,000,000 | |
| 22 | 1913 | | 0 008 | ...... ..... | 930 | 1 | .... | .......... | 11,300,000 | |
| 23 | 1913 | | 0.008 | ........... | 950 | 1 | .... | .......... | 9,750,000 | |
| 24 | 1913 | | 0.008 | ........... | 1,000 | ... | ..... | Heated 6 times to 1,000. | 26,000,000 | |
| 25 | 1913 | | 0 008 | .......... | 915–930 | .... | 1 | ........... | 4,300,000 | |
| 26 | 1913 | | 0 008 | ......... | 950 | 1 | .... | ........... | Coarse | |
| 27 | 1913 | | 0.008 | Coarsened in 1 hr. at 950°, No. 26. | 950 | 21 | .... | ........... | Fine | |
| 28 | 1913 | | 0.008 | Refined in 21 hr. at 950°, No. 27. | 950 | 1 | .... | ........... | Coarse | |
| 29 | 1917 | Ruder.... | | Later refined by heating at 1250–1300°. see No. 14. | 1,000 | 3 | .... | ........... | 1,500,000[3] | |
| GROUP III. COARSENING OF FERRITE BY THE COMPETITIVE PRINCIPLE | | | | | | | | | | |
| 30 | 1917 | Ruder.... | Silicon steel | ........... | 1,300 | .... | .... | Fed slowly past 1,300° | 9,500,000,000 nearly | |

grain size is wholly irrelated to that of the pre-existing ferrite, but rather that the grain-size relation of parent to offspring is far closer in the austenite-ferrite or Ar than in the ferrite-austenite or Ac transformation.

It is indeed surprising that Mr. Ruder's electrolytic specimen should undergo this enormous natal coarsening, because its thickness was probably far beyond the limiting thickness, about 0.012 in. (0.3 mm.), which Stead and Carpenter found sufficient to bar natal coarsening. His coarse grains have not only their size but also their ragged outline and irregular shape to show that they are to be classed with the natally coarsened grains of Stead and Carpenter. They are shown thus not only by their size but by their shape and outline to be formed natally and not inherited from the mother austenite.

[3] This is the size of one grain not the average grain size.

## Discussion

W. E. Ruder, Schenectady, N. Y. (written discussion*).—Professor Howe, with characteristic thoroughness, has demonstrated that Professor Jeffries' "Reversed Inheritance" explanation of the case in question is contrary to fact, as shown by many previous observers.

Since the photographs were my own, I feel it my duty to say a few words in discussion. In the first place, it is to be noted that these photographs were of pressed powders, and although many of Professor Jeffries' original experiments were made from this kind of material, the nature of such a specimen must of necessity be different from that of the fused metal, although the same basic laws of grain growth may apply in both cases, certain factors, such as the rate of change, for example, are undoubtedly different in each case.

That the 1000° C. grain is a natal grain is evident from its appearance contrasted with the more perfectly formed, though smaller, grains in the 1300° C. photograph.

I am not sure that Professor Howe's explanation of the formation of this grain is necessarily the correct one, although it agrees with Stead and Carpenter's results. I am inclined to think that this grain may have been formed on heating, because I have been able to produce grains of the shape and magnitude of Stead and Carpenter's grains by heating to 850° C. pure electrolytic iron, which has been subjected to a strain gradient.

On reheating to 1300° C., the mobility being so much greater, the individual crystals had time for unhampered re-arrangement and growth after their second refinement on passing $Ac_3$, so that we have in this case a true fused metal condition in which the individual crystals are arranged in true equiaxed grains of a size consistent with their high temperature anneal. In this case the grains are equiaxed, inheriting none of the columnar tendencies of the previous grain, because they were not affected by any strain gradient, except possibly some accidental local gradients.

It seems to me that the introduction of pressed powders into the discussion of grain growth, where allotropic changes take place, unnecessarily complicates an already complicated problem.

The accompanying photographs are taken from some work that I have been doing upon the growth of grains in pure and commercially pure metals. These strips of American ingot iron (0.020 in.) have been put through a cycle of annealing temperatures to determine the effect of a previous grain size upon a subsequent structure. The strips were cut wedge shape and stretched to 5 per cent. permanent elongation. The annealing in each case was done by slowly heating in an electric furnace to temperature, holding for one hour and then allowing to cool with the furnace.

* Received Oct. 11, 1917.

The photographs (full size) show that the material after 1000° C. heating gives much larger grains on an originally large-grained material than on originally small-grained material. A subsequent heating of both to 950° C. (no photograph) equalized the structure, however. Both were then about the same size as Fig. 1. This shows that the austenite grains inherited to a certain extent the size of the ferrite grains, at least that they are related to them.

FIG. 1.

FIG. 2.

FIG. 3.

FIG. 4.

Fig. 2 and 4 are of two strips having a uniform strain gradient and heated respectively to 1300° C. and 1000° C. In Fig. 4 the selective grain growth structure induced by the strain gradient has been wiped out on passing through the critical range and a new and more uniform growth (austenitic) has taken its place. In Fig. 2 (1300° C.) this growth has continued. In both of these we have a clear case of inheritance by the ferrite, of the austenite grain size.

These two strips were then reannealed under the same condition as previously, except that the temperatures were reversed. The results

are shown in Fig. 1 and Fig. 3. Fig. 3 (sample 4, reannealed 1300° C.) shows practically the same structure as Fig. 2, as was to be expected, both having started from comparatively fine-grained material and annealed to the same temperature for an equal length of time. Fig. 1 (sample 2, reannealed 1000° C.) however, originally large-grained material, shows a decidedly different structure from Fig. 4 which was treated in the same way but was originally fine-grained. The inference is that the large-grained material either did not refine on passing $Ac_3$, to the same extent as the fine-grained, or that there is a relation or inheritance, if you please, by the austenite from the ferrite.

# Some Unusual Features in the Microstructure of Wrought Iron

BY HENRY S. RAWDON,* WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

THE structure of wrought iron as usually described by metallographists and workers in metal in general is that of a fairly pure iron. Impurities, if present, are usually considered as being in solid solution in the crystals of the ferrite matrix or as forming part of the ever present "slag streaks." Attention is herein directed to one type of these dissolved impurities; to the detection of such impurities, particularly in low-grade irons, and to their possible influence on the physical properties of the metal. The attention of the Bureau of Standards was first directed to wrought iron of the peculiar and unusual characteristics to be described later, in material which had failed in service and was submitted for test. The examination was extended to other grades of wrought iron, to see whether such features are of common occurrence in this class of material.

## MICROSTRUCTURE OF WROUGHT IRON

### *Usual Structure*

The general microstructure of wrought iron is so well known that a detailed description of it here is needless. There is shown in Fig. 2 the structure of a sample of Swedish iron which illustrates well the matrix of ferrite crystals in which are embedded the slag threads so characteristic of the puddling process and the subsequent working of the material. The usual commercial product contains varying amounts of the impurities commonly associated with iron and steel, *i.e.*, manganese, phosphorus, silicon, and sulphur together with small amounts of carbon. Of these, the larger part of the phosphorus and some of the silicon present are held in solid solution in the ferrite while most of the remaining impurities, other than carbon, exist in the inclosures of slag. Aside from the discontinuities introduced by the slag streaks the ferrite matrix has the microstructure and appearance of a pure metal. No definite orientation of the various crystals or grains is apparent; deep etching of the specimens reveals no intracrystalline features (*i.e.*, within the grains) other than those noted in "pure" metals in general (etching pits).

* Associate Physicist (Metallography), U. S. Bureau of Standards.

*Unusual Features*

The unusual features noted may be best illustrated by a brief description of the structure of the specimen in which they were first observed in abundance. This piece was a wrought iron eye-bar, a tension member of a railway bridge which after about 30 years' service was modified to suit the increased traffic. After this modification, the member referred to above failed. The microscopic examination of the metal

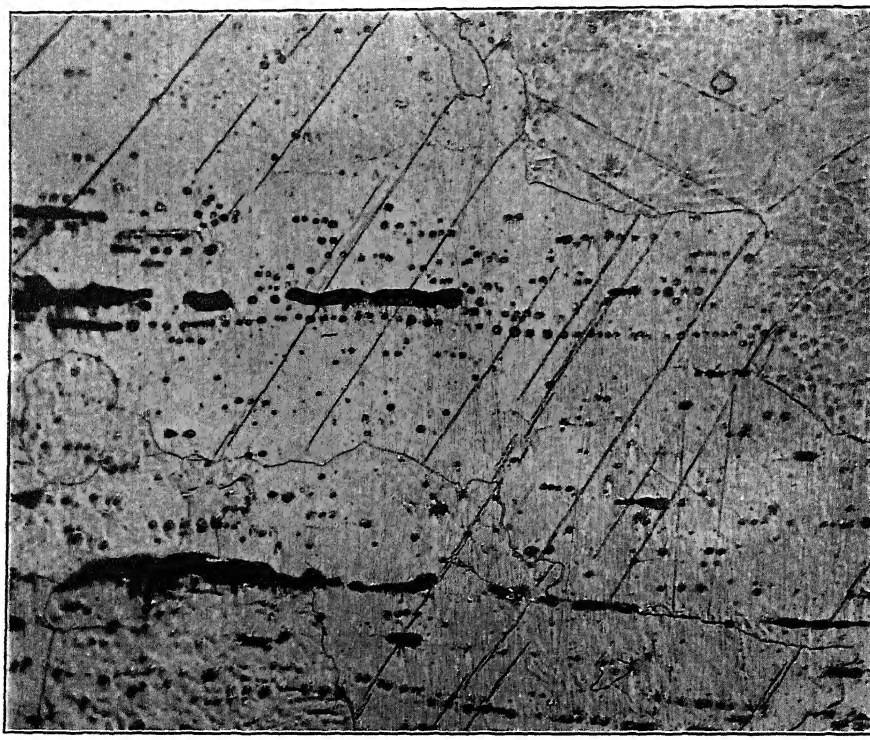

FIG. 1.—THE DARK LINES, PARALLEL WITHIN ANY SINGLE GRAIN, ARE THE NEUMANN LINES. THE MOTTLED APPEARANCE OF THE CORNER GRAINS INDICATES THE NON-HOMOGENEITY OF THE FERRITE DUE TO THE UNEQUAL DISTRIBUTION OF THE PHOSPHORUS. SPECIMEN: W1 (TABLE 1), LONGITUDINAL SECTION. ETCHING, 10 PER CENT. ALCOHOLIC NITRIC ACID. × 100.

revealed, in addition to the usual structure of wrought iron, several structural features so striking as to mark the sample at once as a wrought iron of very unusual properties and composition.

The ferrite crystals presented a peculiar mottled appearance, particularly after prolonged etching with an acid reagent. This etch pattern was not found over the entire surface of the specimen but was restricted to certain streaks throughout the metal. Particularly was it found associated with crystals unusually large in size (Fig. 5). A series of

measurements upon those portions of the material in which such etch patterns were developed in abundance showed the crystals to be as large as 0.25 by 2.00 mm.; a similar set of measurements upon the crystals of the material of Fig. 2 showed the average size to be 0.144 by 0.133 mm. This latter set may be taken as quite characteristic of the grain size of good grades of wrought iron.

The etch pattern referred to above is illustrated in Figs. 1, 3, 4, 5. By using the copper chloride etching reagent referred to later, these patterns may be developed in a very striking manner. They sometimes consist of broad parallel bands, as seen in cross-section, extending across the elongated crystals (Fig. 3) and bearing some resemblance to twinned

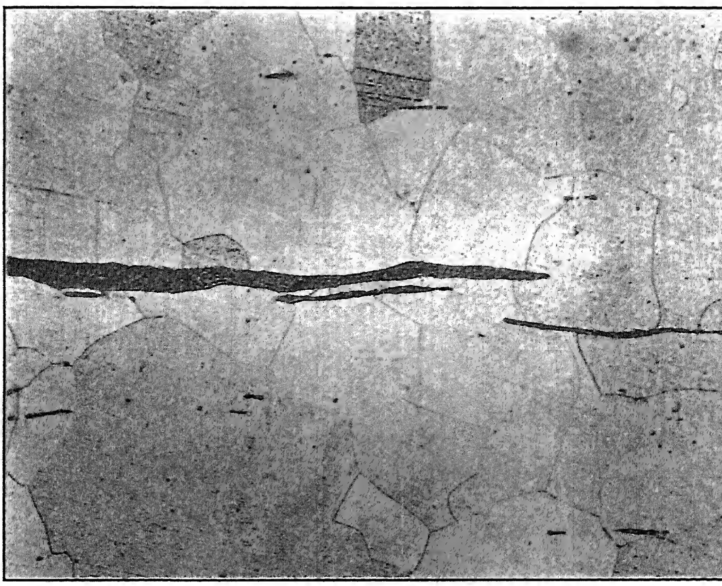

FIG. 2.—MICROSTRUCTURE OF A LONGITUDINAL SECTION OF SWEDISH IRON. ETCHING, 2 PER CENT. NITRIC ACID. × 100.

crystals. More often they present an indefinite mottled appearance of light and dark areas very similar to the shadows cast by the sunlight streaming through between the leaves of a tree (Figs. 1 and 5). The crystals of ordinary wrought iron will not exhibit such etch patterns even after very prolonged etching.

The examination of the metal close up to the fracture, which occurred during the service of the material, shows that the break occurred through the crystals and parallel to the markings constituting the mottled etch-pattern at that point (Fig. 14). Many of the crystals close to the face of the fracture show another variety of intracrystalline markings. On casual examination these may be mistaken for scratches left by poor

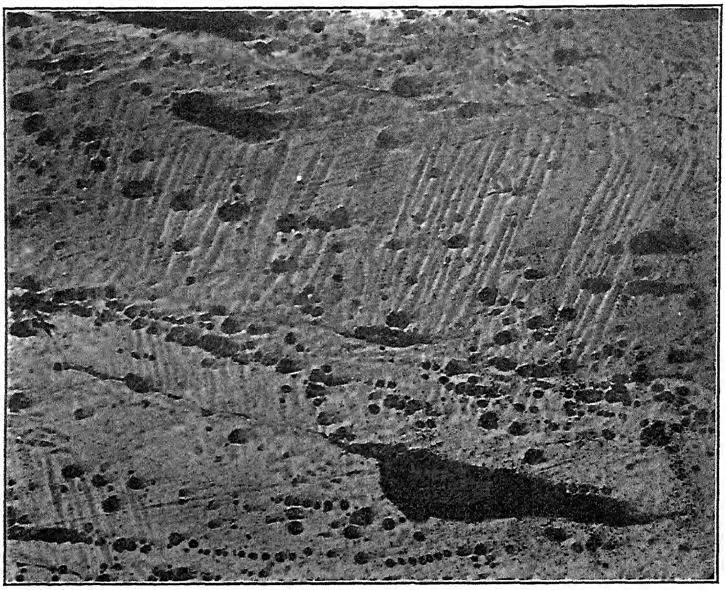

FIG. 3.—SPECIMEN SHOWING THE MOTTLED ETCH PATTERN OCCURRING AS PARALLEL TRANSVERSE BANDS WITHIN THE GRAINS. ETCHING, 10 PER CENT. NITRIC ACID IN ALCOHOL. × 100.

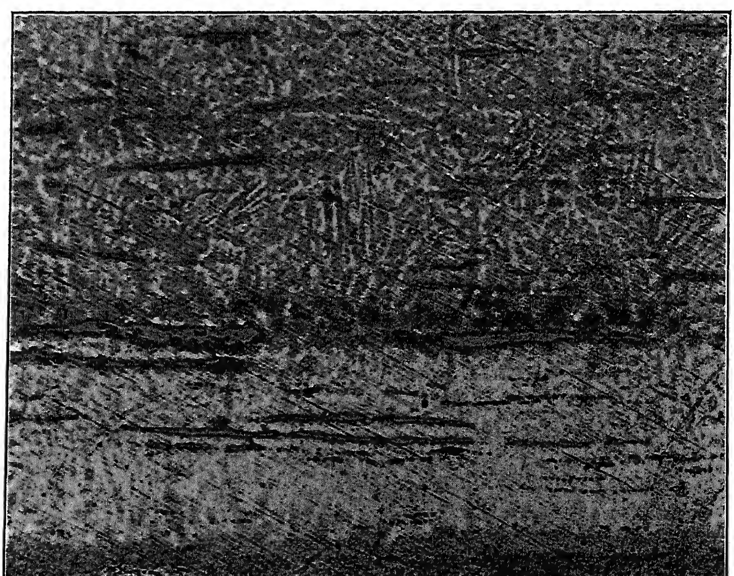

FIG. 4.—WROUGHT IRON PIPE, LONGITUDINAL SECTION. ETCHING, ALCOHOLIC COPPER CHLORIDE ACIDULATED WITH HYDROCHLORIC ACID. (STEAD'S REAGENT.) THE PORTIONS SHOWN ARE FROM THE STREAKS UPON WHICH COPPER DID NOT DEPOSIT. × 100.

polishing of the material (Fig. 1). Closer inspection, however, shows that these markings are parallel to one another within any one crystal and terminate very abruptly at the crystal boundaries. If a slag enclosure lies in the course of one of these markings, the line ends abruptly, reappearing on the other side of the "slag," thus indicating clearly that it is not a scratch. In some crystals, a second set usually not so well developed as the primary ones is seen; the lines comprising this second set are parallel to one another but form a definite angle with those of the first. Crystals showing markings of this general character in considerable numbers usually present a smoother surface and show less of the shadow-

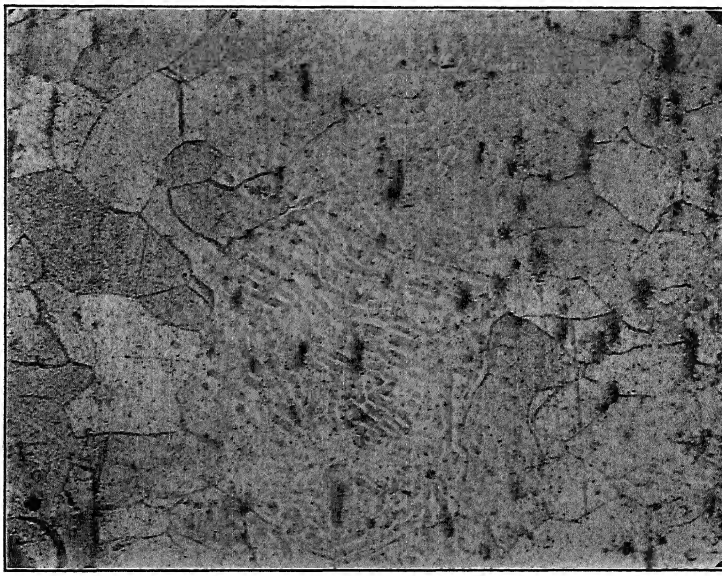

FIG. 5.—THE STREAKS EXHIBITING THE UNUSUAL ETCH PATTERN ARE VERY OFTEN COMPOSED OF GRAINS THAT ARE MUCH LARGER THAN THE AVERAGE FOR THE PIECE. ETCHING, 10 PER CENT. NITRIC ACID. SAME MATERIAL AS FIG. 3. × 100.

like etch-pattern described above than do those crystals in which markings of this class do not occur.

The obvious conclusion is that the non-homogeneity of the individual crystals as indicated by the mottled etch-pattern is to be attributed to some impurity dissolved in the iron but not uniformly diffused throughout the crystal. Robin[1] has called attention to the fact that in ferrite containing considerable phosphorus, *e.g.*, 1 per cent., such a non-homogeneity may exist. Stead,[2] in his extensive studies of the segregation of

[1] F. Robin. *Traité de Metallographie* (Paris, 1912), 184.

[2] *Journal of the Iron & Steel Institute* (1915), **91**, 140. Stead here summarizes his previous work and gives a quite complete bibliography of the literature on phosphorus in iron.

phosphorus in iron and steel, makes but very slight reference to this type of intracrystalline variation of structure which Robin attributes to the non-uniform distribution of phosphorus in the ferrite which forms the body of the crystal. Saklatwalla[3] in his work on the form in which phosphorus occurs in iron, says that up to 1.4 per cent. of phosphorus, a single constituent consisting of the solid solution occurs. Konstantinow's work[4] deals primarily with the alloys high in phosphorus, and those low in this element receive but scant attention.

## Materials Examined

The variations from the usual microstructure of wrought iron, first noted in the wrought-iron bridge member which failed in service, appeared so striking and unusual in character and have been so meagerly described in the literature on the subject that the examination was continued to other grades of iron. From the accumulated series of wrought-iron specimens which have been submitted to the Bureau of Standards from time to time for examination, there were chosen some 35 samples for detailed microscopic examination. Much of this material had been submitted as "unsatisfactory" and one of the purposes of the examination was to see whether such material is characterized by microstructural features similar to those noted above. In Table 1 are listed the materials used. The examination was made by means of Stead's method of etching,[5] making use of an alcoholic cupric-chloride solution acidulated with hydrochloric acid, by which the segregation of phosphorus is shown by the differential precipitation of copper on the surface of the metal. The precipitation of copper on the areas relatively low in phosphorus is much heavier than on the portions of higher phosphorus content, so that the approximate distribution of this element is rendered visible to macroscopic examination. The "high" and "low" phosphorus streaks thus revealed were then examined microscopically to see if the unusual mottled etch-pattern occurred in either.

The results of the examination indicate that the unusual microstructure noted in the failed eye-bar, while not to be regarded as a common feature of wrought iron or always associated with iron of inferior grade, is not unique for the single specimen in which it was first observed. It was noted, however, that such features were found only in material, or portions of material, that, according to Stead,[6] are to be regarded as relatively high in phosphorus.

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[3] *Metallurgie* (1908), **5**, 336, 711.

[4] *Zeitschrift für anorganische Chemie* (1910), **66**, 209.

[5] *Loc. cit.*

[6] *Loc. cit.*

## Nature of the Unusual Features Observed

### *Comparison with Iron-phosphorus Alloys*

For purposes of comparison, to make certain that the unusual microstructure observed is indicative of a non-uniform distribution of phosphorus, a series of iron-phosphorus alloys was prepared. The phosphorus content from 0.014 to 0.48 per cent. for the series of six. Electrolytic iron[7] which had been melted *in vacuo* was used.

Fig. 7 shows the microstructure of an iron-phosphorus alloy thus prepared containing 0.37 per cent. phosphorus. The metal in solidifying

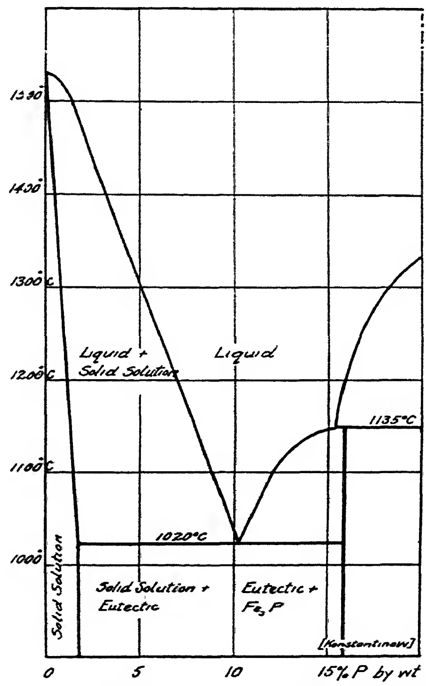


Fig. 6.—Portion of the constitutional diagram of the iron-phosphorus alloys.

freezes selectively, *i.e.*, the solid solution which separates first from the melt is relatively low in phosphorus and the portions successively added are progressively richer in this element. A eutectic consisting of the solution of phosphorus in iron and iron phosphide, analogous to pearlite in steel, formed out of the portion which solidified last of all.

The alloys containing amounts of phosphorus lower than 0.37 per cent. showed the same general structure as Fig. 8, minus the eutectic.

[7] Cain, Schramm, and Cleaves: *Bureau of Standards Scientific Paper* No. 266. Iron, prepared as here described, was used.

The equilibrium diagram based on Konstantinow's work[8] is shown in Fig. 6. According to this, phosphorus is soluble in iron up to about 1.7 per cent., when the system is in equilibrium. Two of the alloys as cast, containing 0.37 and 0.48 per cent. phosphorus respectively, though far below the limit of solubility, show the heterogeneous character of those above this limit. This heterogeneity of structure in low-phosphorus alloys has not been emphasized by Saklatwalla or Konstantinow in their work on this series. The very slow diffusion of phosphorus in the ferrite is the cause of the heterogeneous structure, a condition very apt to prevail in practice, particularly if the heating and working of the material subsequent to the initial rolling is slighted. On annealing the alloy, the eutectic disappears by the phosphide passing into solution in the ferrite and, if the heating is continued, the crystal becomes homogeneous by the diffusion of the phosphorus in solution. Upon etching no pronounced mottled etch-pattern now appears, only a coloring, yellow or brown, of the surface is seen.

The comparison of the structure of the iron-phosphorus alloys with that observed in the wrought irons examined is instructive and very suggestive as to the nature of the markings there seen. The brown or dark bands and spots forming the mottled etch-patterns are the portions of the ferrite rich in phosphorus and each individual crystal in the streaks of the metal showing such mottled appearance is to be regarded not as a simple entity but as a rather complex aggregate. That the different portions of a single ferrite crystal in such portions vary considerably in their properties is to be inferred from the marked variation in composition.

### *Chemical Composition of Materials Showing Such Features*

The analyses for phosphorus of many of the materials listed (see Table 1) show that, though such unusual features of structure as have been described are invariably associated with irons which are rather high in phosphorus, one cannot predict with certainty their presence from a knowledge of the average phosphorus content alone. Some of the samples, *e.g.*, *W* 25, 31, 32, and 33 though comparable with *W* 11, 12, or 16 in respect to the phosphorus content, showed no traces of these unusual features.

### *Etching of Samples*

Upon continued polishing of the specimens before etching, a faint trace of the markings may be seen and recognized by one after becoming familiar with this type of iron. By acid etching, preferably using a 5 or even a 10 per cent. alcoholic solution of nitric acid, the etch-pattern may be satisfactorily developed. The specimen shows to the eye the brown

[8] *Loc. cit.*

and purple oxide tints similar to those which are often observed when hardened and tempered steels are etched with an acid reagent.

By using the cupric chloride solution described by Stead the heterogeneous structure may be developed much more strikingly than with acid alone. It was found, however, that the very definite procedure as described by Stead is not necessary; the sample may be immersed in an excess of the reagent as is the usual practice in etching. Ordinarily, when wrought iron is etched with this copper chloride reagent, it shows the presence of the streaks rich in phosphorus by the deposition of copper on the purer portions so that the "segregation streaks" appear more lightly colored in contrast to the matrix. On wrought irons high in phosphorus throughout and in the iron-phosphorus alloys of rather high phosphorus content, *e.g.*, 0.37 or 0.48 per cent., such a deposition of copper was not observed even after 15 to 20 min. immersion in the etching reagent. The brown tint which results in such cases is very similar in appearance to that obtained with simple acid etching, though the action is more rapid when the copper chloride is used. This coloration has been attributed by Stead to a much retarded deposition of copper. Several samples of the iron-phosphorus alloys and high-phosphorus layers cut out of wrought iron were etched to a pronounced brown by the copper-chloride reagent by immersing for 20 to 40 min. The darkened surface layer was dissolved off with nitric acid after thoroughly washing the specimen, and the resulting solution showed a faint but clearly perceptible test for copper.

Though the amount of copper precipitated is very slight, the decided increase of contrast in the etched specimen when the acid copper-chloride reagent is used over that obtained with acid alone warrants the conclusion that it is to the copper that the more rapid etching action and the striking contrast produced is to be largely attributed.

The areas constituting the mottled etch-pattern which are darkened are those relatively high in phosphorus, as is plainly indicated in Figs. 7 and 8. The ferrite surrounding the iron-phosphide eutectic and that forming part of the eutectic itself constitute the portions which are most strongly colored and not the less contaminated ferrite which solidified first from the melt. This appears to be in direct opposition to the macroscopic results obtained when a sample of wrought iron is etched, in which case copper is precipitated in relatively large amounts upon the layers low in phosphorus. The precipitation of copper upon the streaks of the comparatively pure material is to be attributed to the electrolytic potential of such material with reference to that of copper. The reverse etching of the metal of high phosphorus content cannot be so easily explained upon this basis. The action of the hydrochloric acid alone in the reagent as used will develop the etch-pattern, although much more slowly than when the copper chloride is present. A trial of the two solutions showed

FIG. 7.—ALLOY CONTAINING 0.37 PER CENT. PHOSPHORUS, IN CAST CONDITION. ETCHING, ALCOHOLIC COPPER CHLORIDE ACIDULATED WITH HYDROCHLORIC ACID. × 25.

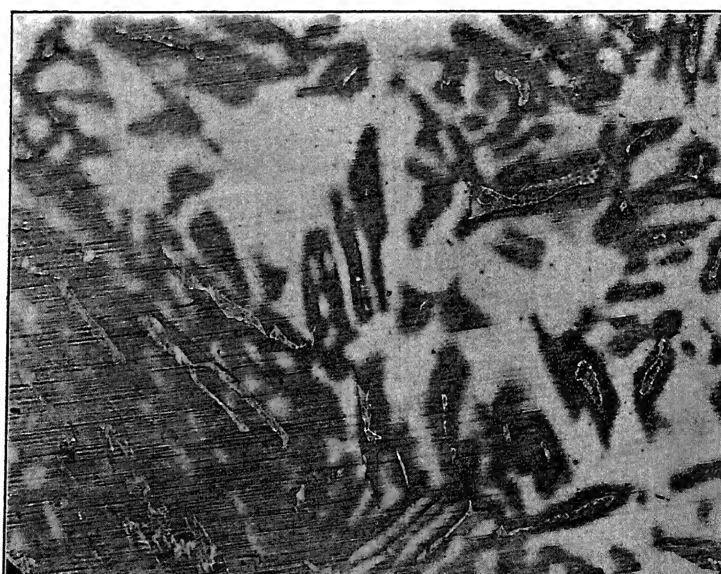

FIG. 8.—SAME MATERIAL AS FIG. 7. SAME ETCHING. THE DARK AREAS ARE PORTIONS OF THE FERRITE HIGH IN PHOSPHORUS. SHOWS SOME OF THE EUTECTIC AS ISLANDS IN THE MIDST OF THE DARK AREAS. × 100.

that by a 12-min. immersion in 1 per cent. alcoholic hydrochloric acid a sample of the iron-phosphorus alloy was not etched nearly as satisfactorily as by a 2-min. immersion in the same solution containing copper chloride. The suggestion is offered that metallic copper as such is not deposited but that there is formed a closely adhering thin film of copper phosphide on the phosphorus-rich portions and so the result which acid etching alone will produce if enough time is allowed is hastened and intensified. The roughening of the surface by a prolonged acid etching is thus avoided by the use of the copper-chloride solution.

The etching reagent described by Rosenhain and Haughton[9] was tried upon the iron-phosphorus alloys, but with unsatisfactory results. It should be noted that if the specimens are mounted in some kind of a matrix for polishing, as is necessary for wires, sections of thin pipe, etc., a matrix of soft alloy or anything of metal, must be avoided. The potential difference existing between this metal and the embedded specimen is often so much greater than that between the various portions of the sample under examination as to prevent entirely the differential etching desired.

*Behavior upon Heating*

The persistence of the unusual intracrystalline structural features upon heating is remarkable. Fig. 11 shows the appearance of a specimen of the wrought-iron eye-bar previously described after heating for 3 hr. at approximately 600° C. (585°–625°) and then allowed to cool in the furnace. No appreciable changes have resulted by this treatment. A second sample (Fig. 12), heated for about 1½ hr. at approximately 725° C. (718°–735°) and furnace-cooled, still shows faint traces remaining of the former condition. Fig. 13 shows the structure resulting from heating a wrought iron high in phosphorus for 1½ hr. at approximately 700° C.; the specimen being allowed to cool in the furnace. The specimen had the structure shown in Figs. 9 and 10, before heating. The eutectic disappeared by solution of the compound, $Fe_3P$, in the ferrite but the nonhomogeneity of structure as shown by the dendritic pattern still persists. This illustrates well the remarkably slow rate of diffusion within the ferrite matrix by which equilibrium is finally attained. The presence of the compound $Fe_3P$ which had separated out from the matrix most probably accounts for the much slower rate of diffusion in this latter case than was observed in the case of the first wrought-iron samples that were annealed.

The slow rate of diffusion of phosphorus in ferrite has often been remarked and it is undoubtedly due to this cause that the mottled structure persists and is not wiped out during the manufacture of the wrought iron, *i.e.*, during the heating, rolling, and forging necessary before the wrought iron reaches the finished condition.

[9] W Rosenhain and J. L. Haughton: *Journal of the Iron & Steel Institute* (1914, No. 1), 515.

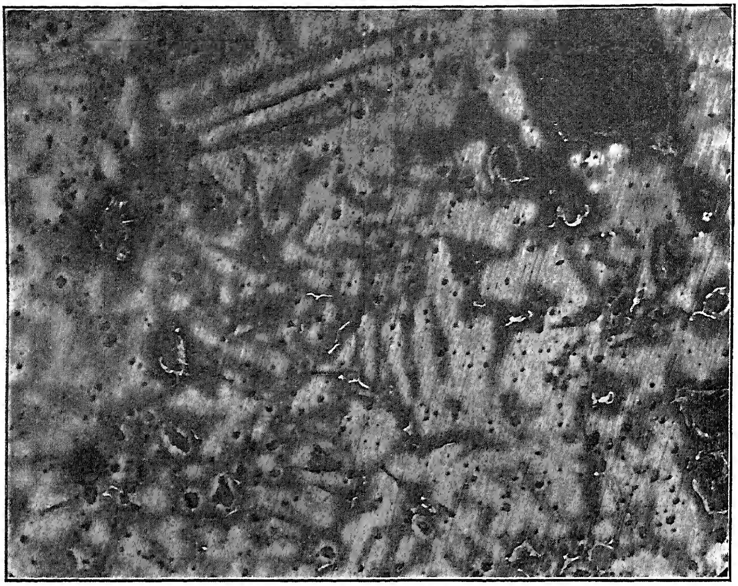

FIG. 9.—HIGH PHOSPHORUS WROUGHT IRON. IN PORTIONS OF THE MATERIAL, ISLANDS OF IRON PHOSPHIDE ARE FOUND; SUCH ISLANDS OCCUR IN THE CENTER OF THE AREAS WHICH BECOME BROWN UPON ETCHING. × 100.

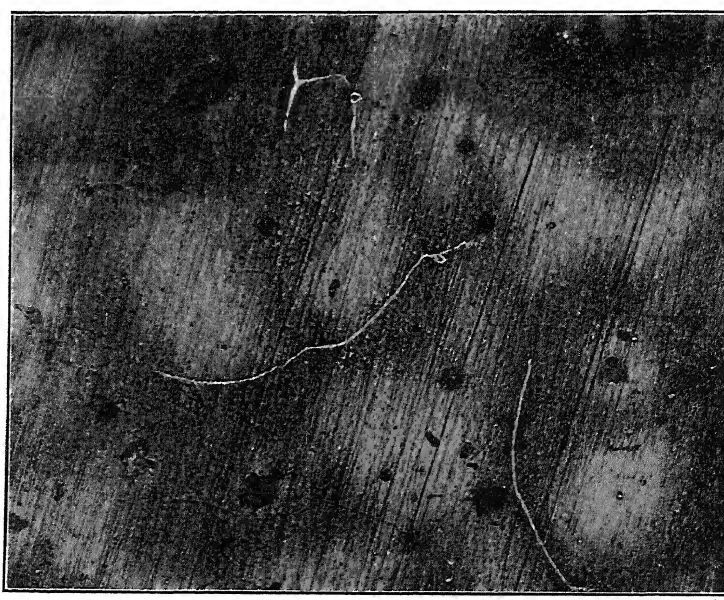

FIG. 10.—SAME MATERIAL AS FIG. 9. ETCHING IN BOTH CASES, ALCOHOLIC COPPER CHLORIDE CONTAINING HYDROCHLORIC ACID. × 500.

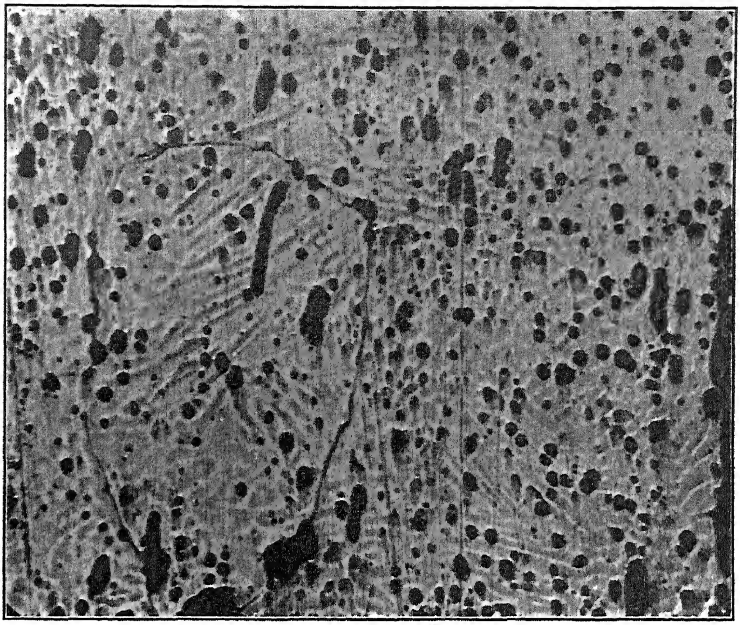

Fig. 11.—Effect of annealing. Material of Fig. 3 heated for 3 hr. at 600° C Etching, 10 per cent. nitric acid. × 100.

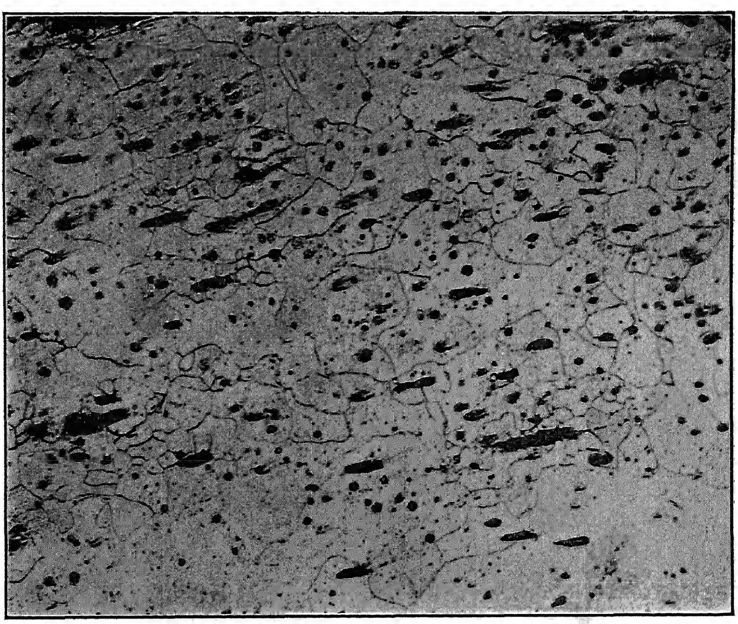

Fig. 12.—Same material as Fig. 11, heated for 1½ hr. at 725° C. Etching 10 per cent. nitric acid. × 100.

## Suggested Significance of Unusual Features of Structure

The significance of the unusual features of microstructure described and their possible relation to the service behavior of such material may be suggested. The occurrence of material of this type in two of the samples examined (*W*1 and *W*37), both of which failed in service and which have the appearance of having failed under the action of alternations or repetitions of stress, suggests a possible relation between this type of structure and the failure of such material.

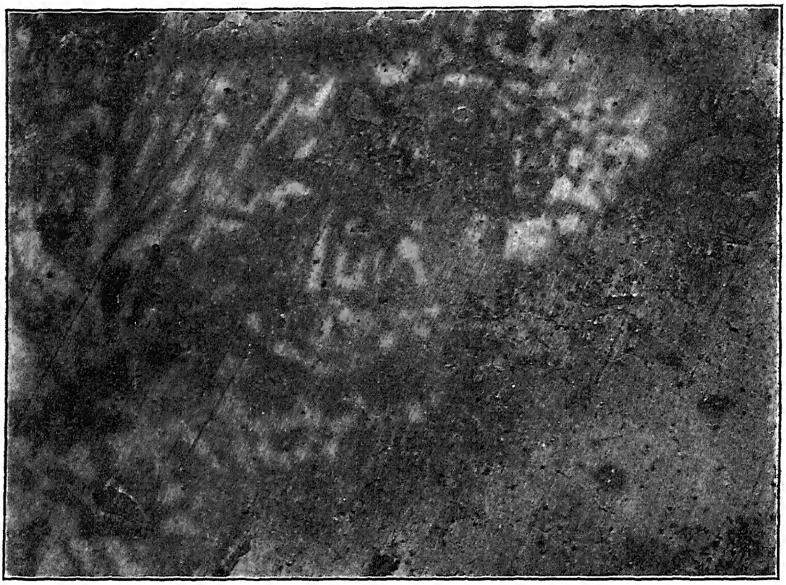

Fig. 13.—High phosphorus wrought iron after annealing. Material of Fig. 9 after heating 1½ hr. at 700° C. The non-homogeneity of structure still persists though the islands of iron phosphide have been dissolved. Etching, alcoholic copper chloride acidulated with hydrochloric acid. ×100.

The method by which "fatigue" breaks occur in metals by the action of repeated stresses has been clearly and conclusively set forth by Ewing and Humphrey,[10] Rosenhain,[11] and others. The minute back-and-forth slip along certain planes occurring within the crystals if repeated a sufficient number of times becomes a permanent displacement, thus initiating an incipient fracture within the crystal. The combined effect of this action within a number of neighboring crystals at some portion of the specimen will be sufficient to cause a real fracture to start at that point if the application of the stresses is continued.

[10] J. A. Ewing and J. C. W. Humphrey: *Philosophical Transactions, Royal Society of London* (1902), **200A**, 241.

[11] W. Rosenhain: *Introduction to the Study of Physical Metallurgy,* Chapter VIII, New York, 1915.

The brittle character of ferrite containing considerable phosphorus is well known. Crystals which show the heterogeneity caused by high- and low-phosphorus bands in juxtaposition should be much more easily "fatigued" by repeated stresses and show a permanent slip much more quickly than crystals which are more uniform throughout in their structure. In particular, this should be true if the bands are transverse to the direction of the stresses acting. The observations upon the fracture of the broken eye-bar appear to confirm this. Though the evidence shown by the failed member is only "circumstantial," as is always the case in the examination of "metal failures," still so closely does it corre-

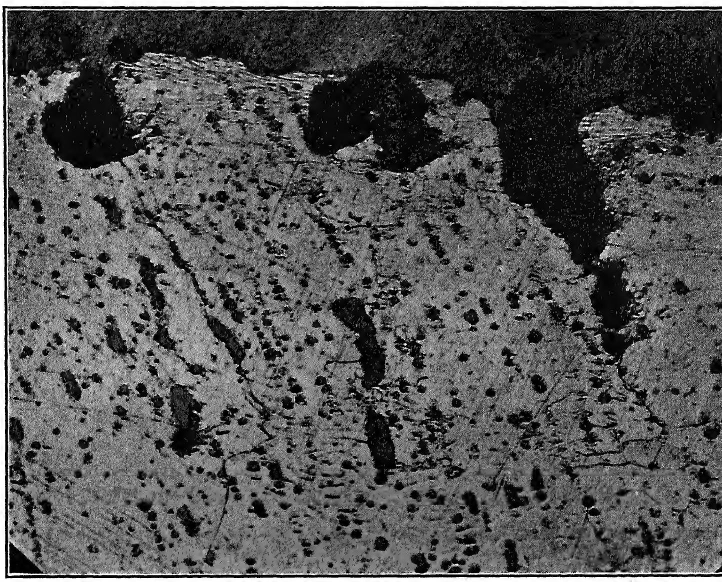

FIG. 14.—RELATION OF STRUCTURAL FEATURES TO THE SERVICE BEHAVIOR OF THE MATERIAL. LONGITUDINAL SECTION OF WROUGHT IRON WHICH FAILED IN SERVICE. THE EDGE OF THE FRACTURE IS PARALLEL TO THE BANDS WHICH CONSTITUTE THE ETCH PATTERN HERE. ETCHING, 10 PER CENT. NITRIC ACID. × 100.

spond to and agree with the results of the work of Ewing and Humphrey that there can be but little doubt as to the nature of the fracture.

The fact that the face of the fracture followed and its course apparently was determined by the bands within the ferrite crystals has already been referred to. Fig. 14 shows that the break occurred parallel to the bands of high-phosphorus ferrite which give rise to the peculiar etch-pattern. Examination of the metal immediately back of the face of the fracture reveals further evidence. Fig 15 shows the appearance of some of the deepest of the etch bands in the crystals close to the break. These, undoubtedly, represent a stage in which minute transverse cracks, *i.e.*, transverse to the direction of the stresses acting, have opened up

within the body of the crystal after a sufficient number of repetitions or reversals of stress. The intracrystalline markings obtained by Ewing and Humphrey by subjecting wrought-iron specimens to repeated alternations of stress in the Wöhler test appear in all respects identical with those shown in Fig. 15.

H. H. Campbell[12] quotes the statement that phosphorus up to 0.20 per cent. is not injurious in wrought iron. This should be taken, however, as referring to an average content of this amount, uniformly diffused throughout the metal. The microsegregation of this element may result in the amount in certain streaks being considerably in excess of

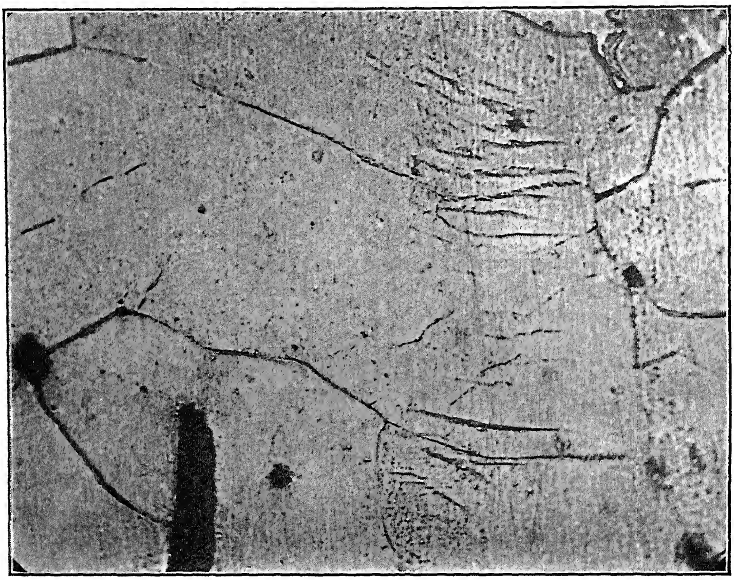

FIG. 15.—APPEARANCE OF VERY PERSISTENT ETCH LINES JUST BACK OF THE FRACTURE IN MATERIAL OF FIG. 14. THE SPECIMEN HAS BEEN TREATED 3 HR. AT 600° C. ETCHING, 10 PER CENT. NITRIC ACID. × 400.

this quantity, while within the individual crystals of such streaks the non-homogeneity with respect to phosphorus is still further accentuated. In the material of Fig. 8 the ferrite immediately surrounding the eutectic has a phosphorus content of approximately 1.7 per cent., or nearly five times the average percentage of phosphorus of the sample. It appears very probable then that in such non-homogeneous crystals as are shown in Figs. 1, 3, 4, 5, 9 and 10, the phosphorus content of the bands may be as much as four or five times the average of the whole. In exceptional cases, portions of a sample high in phosphorus may show traces of iron phosphide that has crystallized out directly from the melt (Figs. 9

[12] H. H. Campbell: *The Manufacture and Properties of Steel*, 91, New York, 1896.

and 10). The metal at such points has a phosphorus content not far below 1.7 per cent., as indicated in Fig. 6.

The straight lines, resembling polishing scratches, shown by some of the crystals near the fracture (Fig. 1), are the well-known Neumann lines. Such markings are usually considered to be "mechanical" twin crystals; *i.e.*, the twinned position of the metal within each of the narrow zones having been brought about by mechanical causes alone.[13] They, in all probability, are the result of shock which finally caused the fracture of the bar. A comparison of these lines with the long narrow bands which often constitute the mottled etch-pattern shows immediately that such etch bands cannot be attributed to twinning or a similar cause.

## VI. Summary

1. Wrought irons high in phosphorus sometimes show a peculiar mottled or banded intracrystalline pattern, which by comparison with alloys of pure iron and phosphorus is shown to be due to a non-homogeneous diffusion of the phosphorus.

2. The examination of a wrought-iron member exhibiting such an unusual structure showed that the break which occurred in service bore a definite relation to such banded markings and apparently was largely determined by them.

3. By the non-homogeneity in the distribution of phosphorus throughout the ferrite crystals, the ill effects of phosphorus may be much enhanced.

4. The examination of a series of wrought irons showed that such features are not to be regarded as common. Many poor grades of iron may be unsuitable for other reasons.

5. A second type of markings is described and illustrated—the well-known Neumann lines—and shown to bear no apparent relation to the other variations in structure described.

[13] H. M. Howe and A. G. Levy: *Trans.* (1915), **51**, 891.

TABLE 1.—*Materials Examined*

| Sample No. | Type of Material | Macroscopic Appearance after Cupric-Chloride Etching | Microstructure (*a*) | Percentage of Phosphorus |
|---|---|---|---|---|
| W 1 | Wrought-iron eye-bar, which failed in service | No copper deposited, surface becomes brown | Very prominent mottled streaks. | 0 356 |
| W 2 | Wrought-iron pipe | Copper deposited uniformly over surface. | No mottled areas seen. The metal is to be classed as steel rather than wrought iron. | |
| W 3 | Wrought-iron pipe | Copper deposited uniformly over surface. | No mottled areas seen. The metal is to be classed as steel rather than wrought iron. | |
| W 4 | Wrought-iron pipe. | Isolated streaks were copper-coated, remainder brown. | Very decided mottled structure in the brown streaks | |
| W 5 | Wrought-iron pipe. | Outer layers slightly copper-coated, central ones slightly brown. | Mottled streaks very prominent in central portion. | {0.130 (*b*)<br>0.200 |
| W 6 | Wrought-iron pipe. | No copper deposited, surface rather brown. | Mottled streaks very prominent in central portion. | |
| W 7 | Wrought-iron pipe. | No copper deposited, prominent brown streaks. | Mottled appearance across entire face of the section. | |
| W 8 | Wrought-iron pipe. | No copper deposited, brown coloration of surface. | Mottled appearance across entire face of the section. | {0 264 (*b*)<br>0 271 |
| W 9 | Wrought-iron "welding" wire. | Copper deposited over all the section. | No mottled areas seen. | 0 059 |
| W10 | Wrought-iron (?) bar. | Copper deposited over all the section. | No mottled areas seen. The material is really a low-carbon steel. | |
| W11 | Wrought-iron pipe. | No copper deposited, central streaks very brown. | Mottled appearance of central streaks very conspicuous. | {0 146 (*b*)<br>0 156 |
| W12 | Wrought-iron pipe. | Copper precipitated on outer layers, brown streaks in central part. | No mottled areas found. | 0.123 |
| W13 | Wrought-iron welding wire. | Uniform precipitation of copper over the section. | No mottled areas found. | |
| W14 | Wrought-iron welding wire. | Inconspicuous streaks in central portion, otherwise quite uniform precipitation of copper. | No mottled areas found. | |
| W15 | Wrought-iron welding wire. | Inconspicuous streaks in central portion, otherwise quite uniform precipitation of copper. | No mottled areas found. | |
| W16 | Wrought-iron pipe. | No copper precipitated, dark central streaks. | Streaks of mottled structure abundant across the section. | 0.160 |
| W17 | Wrought-iron stay bolt | Some light areas showing method of fagotting, precipitation of copper quite uniform. | Very slight trace only of mottled areas. | |
| W18 | Wrought-iron stay bolt | Some light areas showing method of fagotting, precipitation of copper quite uniform. | Very slight trace only of mottled areas. | |
| W19 | Wrought-iron welding wire. | Copper uniformly precipitated over surface. | No trace of mottled structure found. | |

TABLE 1.—(*Continued*)

| Sample No. | Type of Material | Macroscopic Appearance after Cupric-Chloride Etching | Microstructure (a) | Percentage of Phosphorus |
|---|---|---|---|---|
| W20 | Wrought-iron boiler tube which corroded badly in service. | Copper uniformly precipitated over surface. | No trace of mottled structure found. | 0 033 |
| W21 | Wrought-iron boiler tube which corroded badly in service | Copper uniformly precipitated over surface. | No trace of mottled structure found. | |
| W22 | Low-carbon steel pipe | Central streaks free from copper. | Mottled structure very evident in central layers. | |
| W23 | Wrought-iron "goose-neck" | Copper deposited quite uniformly over surface. | No mottled areas found, the metal resembles low-carbon steel rather than wrought iron. | 0 044 |
| W24 | ........... | About 0.7 copper- covered. | ........... | 0.055 |
| W25 | ........... | About 0 1 reddened by copper, other portions show prominent brown streaks. | ........... | 0 136 |
| W26 | ........... | About 0.9 copper-covered. | ........... | 0 079 |
| W27 | ........... | About 0.7 copper-covered, with outer portion clear. | ........... | 0.059 |
| W28 | ........... | About 0.9 copper-covered. | ........... | 0.073 |
| W29 | ........... | Isolated copper-covered streaks aggregating 0.3 total area, other portions clear or brown. | ........... | 0.112 |
| W30 | Wrought-iron round stock for forging purposes (chains, etc.) All but W 28 and W 34 were reported to give unsatisfactory results in the preliminary tests of the material. | Surface clear with brown streaks with one small central copper-covered streak. | No mottled areas were found even in those showing but little precipation of copper, except Nos. 29, 30, and 36, which show a slight trace. | 0.099 |
| W31 | ........... | Only a few narrow streaks are copper-colored, about 0.1 of area. | | 0.115 |
| W32 | ........... | 0.1 copper-covered, remainder clear with brown streaks. | | 0.118 |
| W33 | ........... | One fine streak only, copper-coated, prominent dark streaks in remainder. | | 0.125 |
| W34 | ........... | Less than 0.5 is copper-covered, brown streaks in the clear portion. | | 0.115 |
| W35 | ........... | 0.8 to 0.9 of area is copper-covered. | | 0.066 |
| W36 | ........... | 0.6 area copper-coated brown streaks in remainder. | | 0.089 |
| W37 | Link of wrought-iron chain which failed in service. | No copper deposited, surface was colored brown. | The mottled etch pattern is very prominent. | |

(*a*) Under "microstructure" the presence of streaks showing the mottled structure as shown in Figs 3, 4 above is noted, and not a complete account of the microstructure.

(*b*) 1 indicates the entire cross-section: 2 the central layer, approximately one-third, the cross-section.

## Discussion

Henry Fay, Cambridge, Mass. (written discussion*).—Mr. Rawdon has made a distinct contribution to our knowledge of wrought iron, and perhaps of steel. It will undoubtedly lead to more careful scrutiny of high-phosphorus steels, with perhaps important results. In view of his results, it would seem to be unprofitable to push further any propaganda on the use of higher-phosphorus low-carbon steels until further information is available. As is well known, and as Mr. Rawdon clearly shows, the process of solidification is a selective one, and there is abundant opportunity for those portions of the melt last to solidify to become rich in phosphorus, and the diffusion of this phosphorus-rich area is slow. Hot work does not help the situation appreciably and only serves to concentrate the low-melting material. It is undoubtedly true, as some writers have recently claimed, that the specifications for steels as they now exist eliminate some perfectly good material. Higher phosphorus in steel may be perfectly safe provided all of the phosphide is completely and uniformly diffused in the solid solution. If, on the contrary, it is not evenly diffused, and there is a reasonable chance for it not to be, it makes the steel unsafe. Until our knowledge of the rate of diffusion of phosphorus in steel, and the effect of other elements upon the rate, is more complete, it seems to me unwise, in view of Mr. Rawdon's experiments, to listen to the recommendations for the use of higher-phosphorus steels.

Attention was called some years ago[1] to a peculiar case of segregation of phosphorus in a piece of cold-rolled shafting. At the time of the publication of the paper, a review only of Mr. Stead's first paper on iron and phosphorus was available. It was stated in my paper that there existed in the segregated area what appeared to be the eutectic of iron and iron phosphide. It had a distinctly crinkled appearance before and after etching, as Mr. Rawdon describes. The phosphorus content in the segregated zone was 0.21 per cent. This appearance had not been connected with other material as Mr. Rawdon has now done with wrought iron. This same crinkled appearance has frequently been noticed on unetched low-carbon steels, but the meaning of it has not been evident until now.

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* Received Oct. 11, 1917.

[1] Henry Fay: Segregation of Phosphorus in a Piece of Cold-Rolled Shafting. *The Metallographist* (1901), 4, 115.

# The Erosion of Guns

BY HENRY M. HOWE,* LL. D., SC. D., BEDFORD HILLS, N. Y.

(New York Meeting, February, 1918)

## CONTENTS



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1. *Introduction.*—This paper is based in large part on the examination of two rings, shown in section in Fig. 4 to 15, cut from an eroded 14-in. (35.56-cm.) gun liner, and containing, according to Booth, Garrett, & Blair: Carbon, 0.380 per cent.; manganese, 0.655 per cent.; nickel, 2.810 per cent.; vanadium, 0; chromium, a few hundredths of 1 per cent.; tungsten, 0; molybdenum, 0.

These rings were received from Admiral Joseph Strauss, then Chief of Ordnance, U. S. Navy, through the kindness of Mr. Hudson Maxim, Chairman of the Committee on Ordnance and Explosives, of the U. S. Naval Consulting Board.

I also discuss Prof. Fay's interesting paper on this subject.[1]

The gun from which these rings were cut was 642.5 in. (53 ft. 6.5 in.—16.32 m.) long over all, and had been worn out at the proving ground by firing 170 rounds. The rear ring was cut at the origin of rifling, the forward one at 207 in. (17 ft. 3 in.—5.26 m.) from the breech face, or about one-third way from the breech to muzzle.[2] I understand that these guns weigh 63 tons, and fire a shell weighing 1400 lb. with a muzzle velocity of 2600 ft. per second and a muzzle energy of 65,000 ft.-tons.

The erosion has been marked in the forward ring, while in the rear one it has erased the lands almost completely.

A metallurgist in writing about this intricate subject on which he is necessarily inexpert, must appeal to the indulgence of the expert. His excuse for his intrusion must be that through his familiarity with hardening and with the behavior of the micro-constituents of steel under other conditions he may help the expert to a true understanding of these phases of the subject.

A gun is rifled in order to give the projectile such rapid rotation about its own axis that, like a gyroscope, it can resist the tendency to tip sidewise or vertically caused by the greater side wind pressure on its

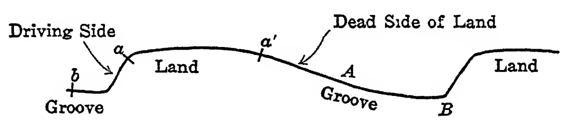


FIG. 1.—DETAIL OF TRANSVERSE SECTION OF THE RIFLING OF RING NO. 2 OF 14-INCH GUN NO. 1 (SEE FIG. 4). *A* AND *B* ARE WHERE FIG. 6 AND 5 WERE TAKEN.

broad base than on its pointed nose, and the tendency to invert and travel butt first which arises if the center of gravity lies behind the center of figure. This tendency we prevent in making an arrow by shoeing its point or feathering its rear. In brief, the rapid rotation makes the motion strictly vectorial.[3]

The rifling consists of a series of "lands" or spiral projections from the bore of the gun. The projectile, through its copper driving band, engages these projections and is thus set spinning clockwise about its axis.

The lands are sometimes rectangular as sketched in Fig. 26, and sometimes, and more rationally, of the shape shown in Fig. 1 and 4.

[1] H. Fay: Erosion of Guns—The Hardening of the Surface. *Trans.* (1916), **56**, 468.

[2] Commandant A. L. Millard: Private communication, Oct. 3, 1917.

[3] Even if the tendency to invert were removed by setting the center of gravity far forward, and if the projectile were symmetrical forward and aft so that a side wind would effect both its ends alike, a wind on the quarter would still tend to turn it.

Two strong agencies tend to erase these projections, and thus to "erode" the bore, the scraping of the driving band and the rush of white-hot gases and perhaps liquids, and of incandescent and yet ungasified explosive. These may for our purpose be grouped together as "the gases." The scraping effect of the driving band is due to its combination of extremely rapid motion with tangential pressure great enough to overcome the resistance of the projectile to receiving a rapid rotation. The seriousness of this scraping effect is easier to conceive than that of the rush of the gases. We may get some notion of this latter from the fact that its eroding effect is greater than that of the driving band (Sec. 47).

That even fluids have great erosive power is shown by the strong cutting action of filtered water escaping from under great pressure, as at leaks in hydraulic apparatus.[4]

That erosion requires dynamic action is shown by its absence from closed bombs "in which both the pressure and the temperature are the same as in guns."[5]

The thickness of the layer eroded in each round from a 12- or a 15-in. gun is about 0.0005 in. (0.0127 mm.) according to Mr. Maxim.[6]

Erosion must needs increase rapidly with the size of the gun and the weight and velocity of the projectile. The heating effect must increase with the length of the gun because this implies a longer exposure to the hot gases. To vary Sir Frederick Bramwell's[7] illustration, to pass from a 6- to a 12-in. gun and to double all linear dimensions would increase the weight of the powder eight fold, while only doubling the inner circumference. The corresponding increase in the weight and velocity of the projectile would increase its torque greatly, and hence the total tangential pressure of the driving band on the lands, while only doubling the circumference available for lands. With even 7-in. muzzle-loading guns of 7 tons weight, with 30 lb. of powder, and a 115-lb. shell, erosion became serious.[8]

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[4] See Sir F. Abel: On the Erosion of Gun-Barrels by Powder Products. Discussion by Col. Dyer, *Journal of the Iron and Steel Institute* (No. 2, 1886), 491, and Action of the Projectile and of the Explosives on the Tubes of Steel Guns. Discussion by E. P. Martin, *id.* (No. 2, 1898), **54**, 241. T. W. Nordenfelt (*id.* (No. 2, 1898), **54**, 239) referred erosion chiefly to leaking gases. He had seen a large gun, after only seven rounds, using "hot" powder and with very high velocity, eroded a finger-thickness deep.

[5] Admiral Ralph Earle: *Trans.* (1916), **56**, 494.

[6] *Op. cit.*, 485. The caliber of the gun is increased about 0 001 in. in each round, which implies the removal of 0 085 in. (2.18 mm ) in the 170 rounds of my two rings. This must evidently be taken as an average quantity, because the erosion is so much more rapid at the breech than at the muzzle.

[7] Sir F. Bramwell: Discussion of Sir F. Abel: On the Erosion of Gun-Barrels by Powder-Products. *Journal of the Iron and Steel Institute* (No. 2, 1886), 488.

[8] *Idem.*, 466.

Erosion is accompanied by the hardening (Sec. 4) and cracking (Sec. 26) of the face of the bore, the main cracks being longitudinal in the grooves and transverse on the lands (Fig. 16).

2. *Definitions.*—Let "outer" and "inner," "outside" and "inside" respectively refer in every case and under all conditions to parts farther from the axis and nearer to it. Thus the outer face of the hardened layer, Fig. 4 to 15, is the face farther from the axis, and its inner face is that nearer the axis, the bore face.

3. *Brevity of the Heating.*—If the velocity of the projectile through the 36 ft. from my forward ring to the muzzle was equal to the muzzle velocity of 2600 ft. per second, then the length of time between its passing that ring and escaping into the air would be 0.014 sec. But as there is very appreciable acceleration during this travel, this period may be taken as of the order of 0.02 sec. This is in rough accord with the data of Mr. Maxim[9] that the projectile stays in a 12-in. 45-caliber* gun a total time of about 0.062 sec., and of Admiral Earle[10] that about half the time is occupied in moving forward the first caliber. The total time of travel for such a 12-in. gun is given by others as 0.0471 sec.

Because the length of time increases roughly with the caliber, these two estimates of the time of travel would correspond to 0.07 and 0.05 sec. respectively for a 14-in. gun.

## I. The Hardening of the Bore

4. *The Hardening of the Bore.*—After even as few as five rounds have been fired, the surface of the bore is found to be hardened. At first this hardened layer is confined to the rear of the gun, where the heating has been longest and highest. As firing progresses, it thickens and extends further forward (Sec. 19 and 20), decreasing in thickness from breech to muzzle, where in the early stages it may be confined to a very thin layer on the driving edge of the lands, as in Prof. Fay's case. The hardened layer is the light narrow band, about ⅛ in. (3 mm.) wide between the steel and the copper plating in Fig. 5 to 7 (see Sec. 32) and in Fig. 12 to 15.

While in my rear ring it is an annulus of nearly uniform thickness, as would be inferred from the almost complete effacement of the lands, in my forward one it is greatly thickened at the salient angle of the driving edge, and thinned correspondingly at the re-entrant angle, Fig. 5.

5. *The thickness of the hardened layer* and its increase from muzzle to rear, are shown in Table 1.

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* That is to say, a gun the length of which is 45 times the diameter of its bore, or "caliber."

[9] *Trans.* (1916), **56**, 485.

[10] *Idem.*, 549.

TABLE 1

| | Thickness of Hardened Layer | |
|---|---|---|
| | Inch | Mm. |
| Prof. Fay's rings from 12-in. gun: | | |
| Muzzle (on driving edge only)... ... ... ........... | 0.0001 | 0.0025 |
| 10 calibers from muzzle........ .... ............... | 0.0004 | 0.010 |
| 20 calibers from muzzle.......... ... ....... ...... | 0.0008 | 0.020 |
| Forcing cone (initial depth of rifling ⅝ normal)....... | 0 0013 | 0.033 |
| Origin of rifling (initial depth of rifling ½ normal) ..... | 0.0015 | 0.038 |
| My rings from 14-in. gun: | | |
| Forward ring marked *Ring No.* 2..................... | 0.0047 | 0.120 |
| Forward ring marked *Ring No.* 2 driving edge......... | 0.0063 | 0.160 |
| Rear ring marked *Ring No.* 1........................ | 0.0055 | 0 140 |

6. *Martensitization.*—That this thin hardened layer consists of martensite has been reported independently by Belaiew and Rosenhain,[11] and has been confirmed by Fay.[12] This martensitization, of course, can come only from rapid cooling from above $Ac_1$, and is more likely to represent rapid cooling from a decidedly higher temperature. In the section cut from a 4.7-in. rapid-firing gun tube examined by Roberts-Austen and Osmond,[13] the martensite could not be definitely identified, but Prof. Fay's identification seems to be complete, first in that his hardened layer was changed into troostite on heating to 300°,[14] and second in that he detected the martensitic structure.[15] Hence it certainly represents the heating of the surface of the bore at least to above $Ac_1$, followed by a cooling so rapid as to catch the metal in the intermediate state of martensite, by denying the time necessary for the transformation in cooling to go farther. This is wholly concordant with the long-known fact that the transformation from the alpha or pearlitic to the gamma or austenitic state is extremely rapid, while the reverse transformation is relatively slow. It is quite parallel to the hardening of the surfaces of rails by the slip of the driving wheel, and to that of the surface of a rolling-mill roll which slips past the surface of a sticker. In each case a thin layer of surface is heated above $Ac_1$, and thus converted at least partly and

[11] *Proceedings,* VI Congress International Association for Testing Materials (1912), **2,** Section A, 127.

[12] *Trans.* (1916), **56,** 474.

[13] Roberts-Austen: *Journal of the Iron and Steel Institute* (No. 2, 1898), **54,** 235.

[14] *Op. cit.,* 475.

[15] *Op. cit.,* 484. In the *U. S. Ordnance Department Tests of Metals* (1913), 137, it is reported that the firing has converted nickel steel into martensite, consisting of "a mass of needle-like crystals interpenetrating one another at an angle of about 60°." Here the hardened layer is 0.015 in. thick. These observations I understand that we owe to Prof. Fay.

probably completely into austenite, and is immediately re-cooled by conduction to the cold mass of outer metal too rapidly to permit the transformation from austenite back toward pearlite and ferrite to go beyond the martensite state.

7. *The Hardening Repetitive.*—In the second round the inner part of the layer hardened in the first round is eroded, while the remainder is again raised above $Ac_1$ and thus briefly annealed, to re-harden almost immediately, as it cools again quickly by the outward conduction of heat. Because part of the initial inner surface was eroded in the first round, $Ac_1$ will travel in the second round farther outward than in the first, by approximately the thickness of the eroded layer. Thus each suc-

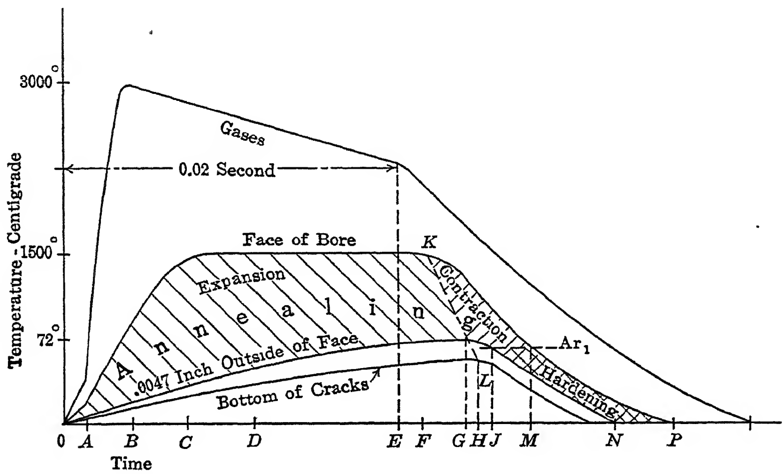


FIG. 2.—THE FIRING CYCLE—A TENTATIVE SKETCH.

cessive round removes by erosion the inner surface of the layer hardened in the preceding one, momentarily anneals and immediately re-hardens the rest of that layer, and restores it approximately to the thickness which it had after the preceding round, by adding about as much to its outside as is removed by erosion from its inside. We shall see that this restoration is only approximate.

8. *The Temperature Cycle.*—Fig. 2 gives the conception which I form, from the scanty data before me, of the cycle of temperature changes at any given section forward of the origin of rifling, for instance, in my forward ring. The actual temperature, of course, varies from place to place, the gas pressure, for instance, being progressively less for any given part of the cycle the farther we pass forward.

In the first period, *OA*, in which the driving band is passing, there should be a material rise of temperature, because of the friction of the band and of the leakage of gas. Yet the quantity of gas compared with the extent of metallic surface with which it is in contact is so small that

the rise of temperature should be much less than in the next period. During this first period the hardened steel lips of each crack-mouth, now wide open, bite off a mouthful of copper from the driving band (Sec. 38).

In the second period, $AB$, which begins as the driving band leaves the section in question, the temperature and pressure of the gas rise to a maximum almost instantaneously, whence the temperature decreases and the pressure drops with the forward motion of the projectile, and with the absorption of heat by the rapidly increasing exposed surface of the bore and by the base of the projectile. This drop accelerates as the projectile escapes from the muzzle, at the end of about 0.02 second.

The solid metal forming the face of the bore rises rapidly to the temperature of erodible mobility, that is, to a temperature so high that the metal becomes soft enough to be swept along by the rush of gases, so that the solid metal is covered with a forward moving layer of very high but indeterminate temperature. Were any one part of it exposed long to the gases at or near 3000° C. it would rise far above the melting point. But this forward moving layer probably turns over rapidly, so that the cooler particles which have just been torn from the solid layer while they were at the temperature of erodible mobility, are quickly mixed up with the far hotter ones which have just been exposed to the hot gases. The motion is that of the thin forward edge of the surf running swiftly landwards on a flat beach under a strong sea-wind, the wind-sped upper layers far outrunning the under ones retarded by contact with the sand (Fig. 28). During the early stages of this forward sweeping, new lots of solid metal are continually transferring themselves to the fluid or pasty moving layer, and during its later stages this transfer is reversed.

Of the metal swept forward a part causes choking of the muzzle end, but much must be swept bodily out of the gun either as a mist or actually volatilized (Sec. 52), because the quantity of metal lost by erosion seems to exceed greatly that which causes choking by being added to the forward parts.

Thus the erosion at any section in any round represents the excess of the metal swept forward thence, plus that entrained thence as mist, plus that volatilized, less that swept forward from the rear and resolidified here.

At the end of the cycle the metal forming the very face of the bore at any given section is that which has been swept forward to this place from the rear parts of the bore in a pasty state, and outside this is the metal which has just failed to reach the temperature of erodible mobility.

Because the solid face of the bore is so much cooler than the gases, it probably continues at its maximum temperature not only after the temperature of the gases has begun to fall between $A$ and $B$, Fig. 2, but perhaps even after that fall has been accelerated at $E$ by the escape of the projectile from the muzzle. The heating of a cooler by a hotter

object should continue even after the hotter object has cooled considerably, provided that it is still materially the hotter of the two.

The outer limit of the hardened layer, that is of the layer 0.0047 in. at the face of the bore, in like manner should continue to grow hotter after that face itself has started to cool, say to time *G*, and the heating of the bottom of the cracks should continue still later, say to time *H*.

The period of heating and hence of expansion of the gases is thus extremely brief, *OA*, while that of the successive layers of the steel is progressively longer as we pass outward from the face of the bore.

The line *KL* sketches tentatively the boundary between the periods of expansion and of contraction of the several layers, the contraction beginning with the beginning of the cooling. The cracks close progressively during the period of expansion, and re-open during that of contraction. The fullest closing of the crack-mouth comes at time *K* and that of the root at time *L*.

The hardening period is of course confined to those layers which rise effectively far above $Ac_1$ as explained in Sec. 9. Each layer enters this hardening period as it cools below $Ar_1$, which is drawn as a horizontal line only for simplicity. In fact, it falls as the temperature reached in the heating, *T* max, rises and as the rate of cooling increases.

During all the heating and during the cooling down to this temperature, the metal is annealing. The annealing and the hardening periods are indicated roughly in Fig. 2. Evidently the hardening period will begin earlier and earlier as we pass outward from the face of the bore. This is equivalent to saying that, in its axisward migration, the $Ar_1$ isotherm arrives earlier at the outer surface of the layer to become hardened than at its inner surface, the bore.

9. *The Hardened Layer is the Merged Layer.*—Starting with the sorbite, or very intimate mixture of microscopic particles of ferrite with cementite, of which the gun liner consists initially, its hardening consists of two wholly distinct necessary steps. First the individual particles of ferrite and cementite merge, on passing above $Ac_1$, to form a single substance, austenite. It is only at temperatures above $Ac_1$ that this merging is possible. Second, during the rapid cooling past $Ar_1$ this austenite transforms into martensite. This martensitization is the essence of hardening, and the hardened layer owes its hardness to its being martensite. Martensite can form only during the rapid cooling of austenite, so that the merging to form austenite is absolutely essential to the hardening. Hence, given the rapid cooling which inevitably occurs in the gun liner, it is only in that layer in which the ferrite and cementite have actually merged and formed austentite that hardening can occur.

10. *The Progressiveness of Merging.*—It is only at the very contact between adjoining particles of ferrite and cementite that this union can begin, quite as a mixture of salt and water can unite only at the surface

of the various grains of salt. The austenite layer thus formed about each grain immediately begins dissolving an additional layer from these same particles of ferrite and cementite, quite as the brine between the grains of a mixture of ice and salt begins dissolving both ice and salt at the surface of the various grains. In effect it is by entering the intervening brine that the salt and ice of the layer which is now dissolving from each grain reach each other and thus react on each other and bring each other to the liquid state.

Under usual industrial conditions of heating, the exposure to temperatures above $Ac_3$ continues first till all the ferrite and all the cementite

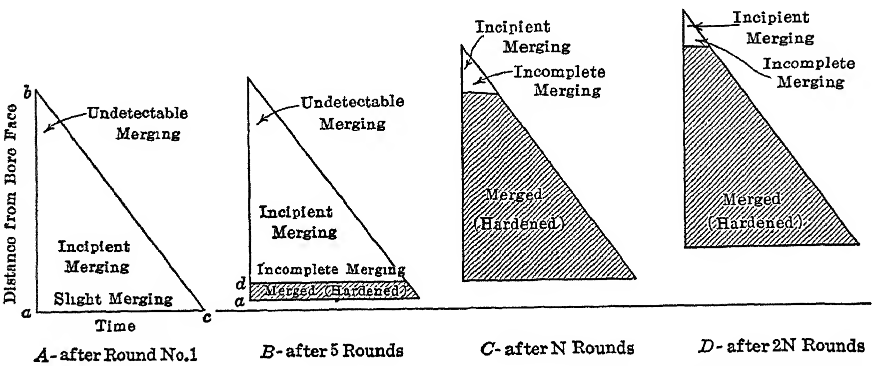


FIG. 3.—HISTORY OF THE EXCURSIONS OF THE $Ac_1$ ISOTHERM, AND OF THE CONSEQUENT PROGRESS OF THE THICKENING OF THE HARDENED LAYER.

have dissolved in each other and merged to form austenite, and second till by diffusion this austenite has equalized its composition. This may be called equalized austenite.

The time needed for this process of diffusion and equalization is denied in the martensitization of the bore of a gun.[16] Even at the very

[16] That a very considerable time is needed to bring about the merging and equalization is indicated by the late results of Portevin (*Revue de Métallurgie* (1916), **13**, 9). In steel hardened by quenching in water under rigorously fixed conditions, he found divers indications of a continuation of this process after the small specimens had reached and stayed for a preliminary fixed period, 2 min., at a fixed temperature, above which they did not rise. Thus (*op. cit.*, 41) steel of 0.25 per cent. of carbon when quenched after 1 hr. at 950° was pure martensite, whereas after 2 min. at 950° its martensite contained spots of troostite. Steel of 1.30 per cent. of carbon (*op. cit.*, 42) contained less pro-eutectoid cementite when its stay at the quenching temperature, 950°, was 20 min. than when it was 2 min. Again, his six steels (*op. cit.*, 49) which had more than 0.30 per cent. of carbon had greater electric resistance when their stay at the quenching temperature, 750°, was 10 or 20 min. than when it was 2 min.

The variations in electric conductivity seem to me too great to be referred to any slight rise of temperature in the central parts of his specimens after the end of his preliminary 2-min. periods, and therefore refer themselves most readily to a continuation of the process of merging and equalization.

bore face the length of time available for merging, that is the time above $A_1$, should be at most but a few hundredths of a second, and this decreases rapidly as we pass outward.

Hence, four results: that at any given section the hardened layer thickens progressively from round to round (Sec. 11); that the rate of thickening decreases asymptotically (Sec. 16); that a troostitic layer forms outside the hardened one (Sec. 17); and that martensitic markings (Sec. 18) are usually lacking.

11. *The progressive thickening of the hardened layer from round to round* is a natural result of the principles set forth in Sec. 9. Let me try to explain by means of Fig. 3 how this thickening occurs. Here ordinates represent the distance outward from the bore face, and abscissæ the length of time that each spot remains above the $A_1$ isotherm, that is, the length of time between the arrival there of the $Ac_1$ isotherm in its outward movement and the return there of the $Ar_1$ isotherm. To fix our ideas, let it be the excursion of the isotherm vertically from the upper side of the bore that we consider. The ordinate of the hatched area represents the thickness of the merged and therefore hardened layer after the fifth round in Diagram *B*, after the *N*th round in Diagram *C*, etc.

Each diagram starts a little higher up on the page than that on its left, to represent the progressive erosion and hence outward movement of the bore face. Moreover, each diagram is a little taller than that on its left, in accordance with the principle explained in Sec. 15. For simplicity we may ignore this in our present discussion.

Diagram *A* indicates that, though even in the first round the $Ac_1$ isotherm travels all the way from *a* to *b*, no considerable hardening occurs, simply because the time during the out and back excursion of $A_1$ is so brief and the mean temperature so low that the resultant merging even in the bore-face layer is too slight. No doubt some merging occurs and is most marked in the layers nearest the bore face. But we may suppose that it is so slight that the proportion of merged austenite to unmerged ferrite and cementite is so small, and the consequent proportion of martensite representing this austenite in the cold steel to ferrite and cementite so slight as to cause no considerable increase in hardness and no appreciable change in the microscopic appearance.

12. *Merging and Hardening are Cumulative.*—Incomplete and even incipient merging are not only true but permanent steps toward complete merging, because so much merging as has occurred in any one layer even in the first round is in large part preserved in the rapid cooling which follows. This is indeed but one aspect of the familiar greater rapidity of the *Ac* transformation in heating up than of the *Ar* transformation in cooling down. Whatever degree of merging is preserved in the cooling is an additional step gained in the following heating up. Because a given layer starts the second round already slightly merged, its merging

goes correspondingly farther in the second than in the first round. The merging gained in the first two rounds is an initial step gained for the third round, which adds a slight further merging to the merging remanent from the second round, and so on.

This explains why hardening of the bore face can be detected after the fifth round, which has added its little merging to the accumulated steps of the first four rounds. This is represented in Diagram *B* of Fig. 3, the hatched area of which represents the layer which has merged so far that while hot it consists substantially of merged austenite, perhaps with a moderate admixture of still unmerged ferrite and cementite in its outer layers. Still outside this is a series of layers less and less fully merged, indicated here as regions of incomplete, of incipient, and of undetectable merging.

The layer *ad* near the bore face in which the merging has approached completion transforms during the rapid cooling into martensite, perhaps with a slight admixture of yet unmerged ferrite and cementite in its outer parts, and thus hardens, so that the hardening, like the merging, is cumulative.

A very thin layer immediately outside this, which is not yet so fully merged that its rapid cooling yields true martensite, in the heating of the sixth round takes the further step in merging needed to convert it substantially into austenite, and hence back in cooling into martensite and thus into the hardened state, thus adding itself to the layer hardened in the fifth round. Immediately outside it is a series of very thin layers each slightly less merged than the one next inside it, and all taking simultaneously in each round a step forward in the merging process.

Thus the spaces occupied by the regions of incomplete, of incipient, and of undetectable merging become gradually pushed out toward the apex of the triangle, and correspondingly thinned, because their collective thickness must needs be contained between the outer limit of the hardened layer and the apex of the triangle, the limit of the outer excursion of $Ac_1$.

If we seek evidence that this progressive thickening which I have thus explained actually occurs, we receive from the literature of the subject the impression that it is so well known as not to need proof. Thus we are told that some hardening occurs even during the first five "proof" rounds.[17] Yet some direct evidence is at hand. For instance, the hardening of the edges of a letter stamped by Prof. Fay on a pressure plug was much thicker after 200 than after 100 rounds.[18]

[17] Roberts-Austen: *Journal of the Iron and Steel Institute* (No. 2, 1898), **54**, 233.
Admiral Ralph Earle: *Trans.* (1916), **56**, 492. "The bores of some guns, returned from proof after five rounds, are found so hard as to resist the ordinary tools used in manufacture."

[18] Fig. 10 and 14, *Op. cit.*, 478, 480.

13. *Three additional causes of the progressive thickening of the hardened layer* may be noted, though their joint effect is probably very slight. Of these, the first is that, as erosion continues, it is accompanied by a leakage of the gases past the projectile, and that because these gases come into such extended contact with the bore walls they ought to give up their heat to them very thoroughly. But whatever gain may occur in the early rounds from this cause is likely to die away as erosion progresses. First, the progressive increase in the erosion brings the leaking gases into increasingly less close contact with the bore walls. Second, this leakage lessens proportionally the high pressure and temperature developed in period *AE* of Fig. 2.

14. *Progressive Roughening of the Bore.*—A second cause might be thought to exist in the progressive roughening of the bore face by its cumulative cracking (Sec. 37), and hence roughening, and in the greater surface for heat-absorption which this roughening might be thought to imply. But this roughening is probably inoperative because, as sketched in Fig. 28, the actual surface of the bore walls is probably covered during this eroding period with a semi-fluid layer, which certainly should cover all the roughness found later in the cold walls, re-cracked in the cooling.

15. *Avoidance of an Endothermic Transformation.*—The third of these additional causes is that the transformation of the initial ferrite plus cementite into austenite in the heating up is endothermic, while the reverse transformation is exothermic. The retention of the hardened layer in the state of martensite at the end of the first round is proof positive that in the cooling of that round the transformation from austenite to ferrite plus cementite was prevented from completing itself, and hence that the system starts from a higher plane of energy in the second round than in the first. In other words, less heat will be consumed in transforming the martensite layer into austenite in the heating of the second round than was consumed in transforming a layer of identical thickness of ferrite plus cementite into austenite in the heating of the first round. This means that in the heating of the second round correspondingly more heat is available for the joint work of heating and transforming other layers. Hence a farther outward travel of all isotherms, including that of $Ac_1$. Because the hardened layer is thicker for this reason at the beginning of the third than at that of the second round, the quantity of heat absorbed in the *Ac* transformation will be correspondingly less and the heat available for heating the metal correspondingly greater. This should favor a further thickening of the hardened layer in the third round, and so forth.

Because the heat absorption in the *Ac* transformation is only a small fraction of the total heat which enters the walls of the bore, the influence of this cause should be small.

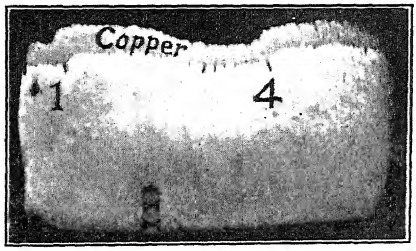


FIG. 4.—TRANSVERSE SECTION. × 1.8.

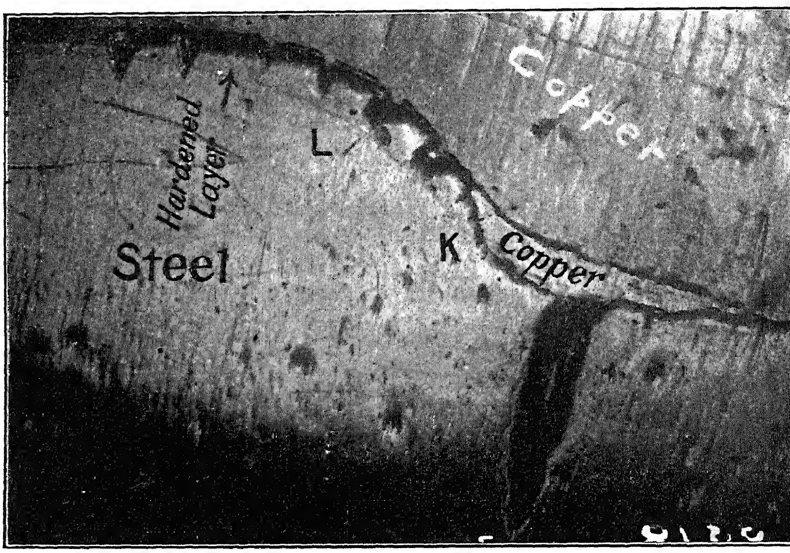


FIG. 5.—DRIVING EDGE. (POSITION $B$ IN FIG. 1.)

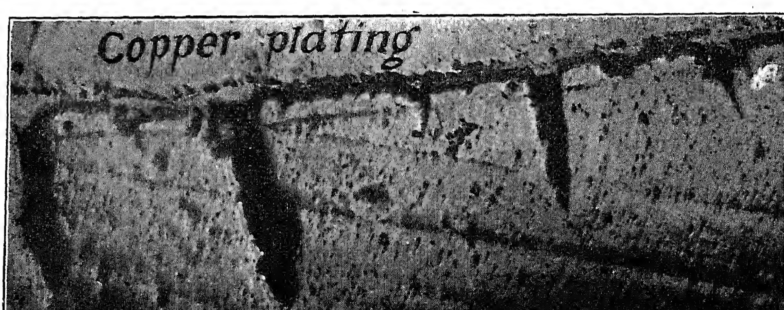


FIG. 6.—DEAD EDGE. (POSITION $A$ IN FIG. 1.)

FIG. 5 AND 6.—TRANSVERSE SECTIONS OF FORWARD RING. × 36.

16. *Asymptotic Retardation of the Thickening of the Hardened Layer.* —Returning to Fig. 3, the condition after a large number, $N$, of rounds may be represented by Diagram $C$. It is probable that during the next $N$ rounds the thickness of the hardened layer increases only extremely slightly, as sketched in Diagram $D$, and that its further increase dies out asymptotically for three reasons.

As we pass from the base to the apex of any of these triangles, the degree of merging decreases rapidly, first because of the decreasing length of the period between the arrival at $Ac_1$ and the return to $Ar_1$, and second because of the decrease in the mean temperature during that period. Thus at the very face the temperature certainly reaches that of erodible mobility, or even perhaps the solidus, whereas half way from $a$ to $b$ it may never reach 1000°.

A third reason is that the hardening can never pass beyond the apex of the triangles of Fig. 3, can never extend to metal which does not rise past

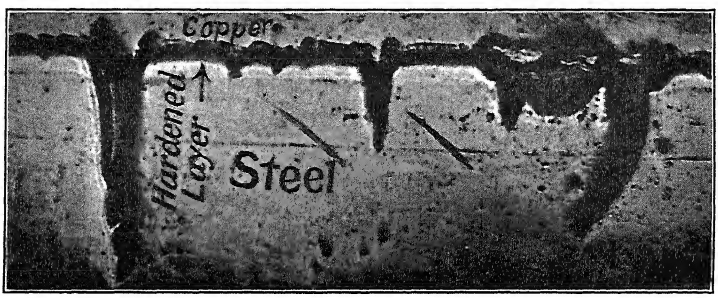


FIG. 7.—LONGITUDINAL SECTION, LANDS. × 36.

$Ac_1$. Indeed, it can never reach the apex, because at a temperature as low as $Ac_1$ a very considerable sojourn is needed to bring about considerable merging. But at every round the base of the triangle is moved outward by the erosion of the bore face, and with it moves the apex into metal as yet absolutely unmerged.

The three supplementary causes of the outward movement of the hardened layer (Sec. 13, 14 and 15) should not prevent this asymptotic retardation of its thickening. The first of these causes loses force as erosion proceeds, the second is probably inoperative, and the third itself dies out asymptotically, because it represents only the addition to the available heat caused by the thickening of the hardened layer, and this thickening itself is dying out asymptotically.

The thickening of the hardened layer in any round represents the excess of the addition to its outside over the simultaneous loss by erosion from its inside. Because of this erosion the merging and hardening cycle starts from a new base in each round, farther outward than in the preceding round by the amount of the erosion. We know that the hard-

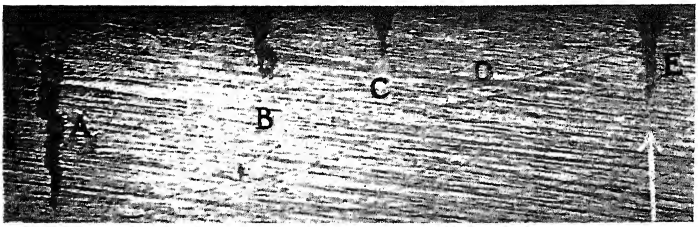


FIG. 8.

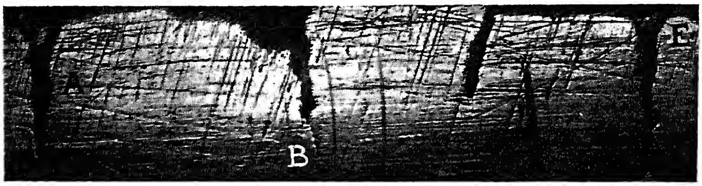


FIG. 9.

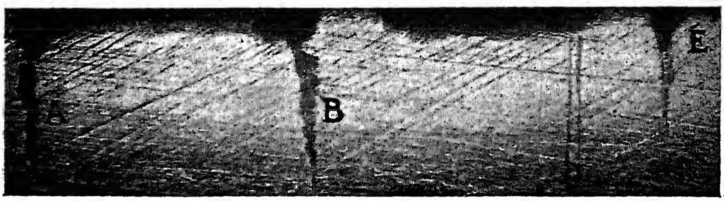


FIG. 10.

THREE SECTIONS ½ MM. APART.

FIG. 11.


FIG. 12.


FIG. 13.


FIG. 14.

LONGITUDINAL SECTIONS, LANDS, FORWARD RING. × 36.

ened layer thickens itself in the early rounds, but we see that this thickening must cease asymptotically, because while the heat evolution remains constant, thus tending to give a constant outward limit to the travel of the $Ac_1$ isotherm, and hence a constant thickness to the merged and hardened layer which can never reach that outer limit, the slight decrease of the heat absorption through the lessening of the endothermic $Ac$ transformation is itself decreasing asymptotically.

17. *The Troostitic Layer.*—These layers of incomplete and incipient merging, consisting as they do of unmerged ferrite and cementite with some martensite resulting from the rapid cooling of so much austenite as the merging has caused, are in effect troostite. It is probably they that form the black line outside of the martensitic layer shown most clearly in Fig. 6 and 14. It owes its blackness in part to its being troostite, which etches black, and in part to its shading off in hardness from the martensite to the unmerged sorbite outside it, and hence becoming oblique in polishing, and hence not reflecting the light back into the microscope.

The facts that in an early stage of erosion the altered layer consists wholly of troostite,[19] whereas in advanced stages, as in my photographs, the

FIG. 15.—LONGITUDINAL SECTION OF GROOVE, FORWARD RING. × 36.

troostitic layer is very thin, and that in at least one other it is hardly detectable,[20] are in harmony with the progressive narrowing of this layer predicted toward the end of Sec. 12. Indeed, in view of the complex conditions attending the formation of martensite and troostite, and especially of the tendency of martensitization to arrest itself sharply by means of the pressure which it causes by its sudden expansion, the troostitic layer might well be lacking under some conditions.

18. *The usual lack of martensitic markings* is a natural result of the conditions set forth in Sec. 9 and 10, for it is hardly to be expected that the normal martensitic structure, which forms in the rapid cooling of equalized austenite, should arise in the rapid cooling either of the fully merged but unequalized austenite which forms the hardened layer or of the complex of unequalized austenite plus unmerged ferrite and cementite which exists outside this. Indeed, typical acicular martensite develops less

[19] Fay: *Trans.* (1916), **56**, 480.

[20] *U. S. Ordnance Department Tests of Metals,* 1913, Photo 8, opp. 135.

readily with the small carbon content of gun-tube steel, about 0.35 to 0.40 per cent., than in higher-carbon steel. In the outer parts of the hardened layer this difficulty is increased by the relatively low temperature to which the metal is heated before the rapid cooling begins. A further difficulty may possibly exist in the extreme rapidity of the cooling. The formation of the typical martensitic structure on a scale large enough to be detected even with great magnification must take time, like every


Fig. 16.—Elevation, forward ring. × 1.8.

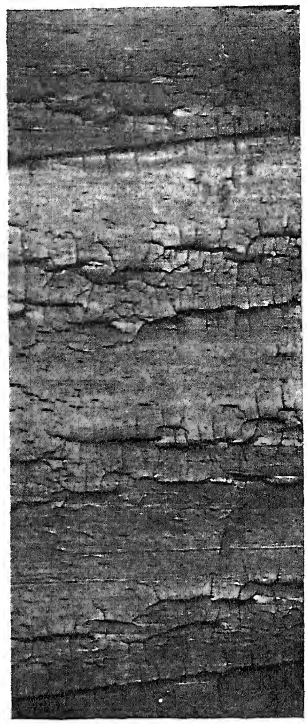


Fig. 17.—Elevation, rear ring. × 1.7.

other crystallization. It is true that time for the development of primary austenite dendrites occurs even when molten iron is poured drop by drop into water. But the rapidity of movement in the change from the molten to the solid phase is likely to be far greater than that in the *Ar* transformation, probably at about 300°, which occurs in the rapid cooling of the gun liner.

19. *The Thickness of the Hardened Layer Should Increase with the Quantity of Heat Taken up by the Walls of the Bore.*—Passing now from the progressive thickening of the hardened layer at any one transverse section to the variations in its thickness from breech to muzzle, let us first note that the thickness of the hardened layer, from 100.00 to 0.0055 in.

(Sec. 5) is of a higher order of magnitude than the thickness of the layer eroded in each round, which is about 0.0005 in., and hence that the velocity of the outward movement of the isotherm of effective hardening is of a higher order than that of the isotherm of erodible mobility. Hence the greater the quantity of heat taken up by the walls of the bore, and the longer consequently the time occupied by the outward travel of these two isotherms, the greater will be the distance by which the outer limit of the migration of the isotherm of effective hardening exceeds that of the isotherm of erodibile mobility, which distance is the thickness of the hardened layer. Another way of expressing this is to say that the therma gradient flattens as the quantity of heat taken up by the walls increases

FIG. 18.—ELEVATION, LAND, FORWARD RING. × 36.

20. *The decrease of the thickness of the hardened layer from breech to muzzle* follows from this last proposition, because the quantity of heat taken up by the walls of the bore must needs decrease as we pass forward because of the falling pressure and hence temperature of the products of combustion, and because of the shortening of the period of high temperature due to the shortening of the time between the passing of the projectile and its escape from the muzzle.

21. *Plastic Deformation as a Contributory Cause of Hardening.*—Prof. Fay, while holding justly that rapid cooling alone can explain the martensitization which accompanies the hardening of the bore, holds that plastic deformation must contribute,[21] because in parts of the metal which have been deformed plastically, either before or during firing, hardening occurs after a smaller number of rounds than elsewhere.

[21] *Op. cit.*, 481.

Unfortunately, his plastic deformation is nearly always accompanied by an increase either of friction or of salience, either of which may well lead to higher heating, and hence to earlier hardening, so that the earlier hardening of the deformed parts is explicable by their higher heating.

Let us now review his evidence briefly. The first of it, that at the muzzle the driving edge of the land only is hardened, is equivocal, because this edge, being harder pressed by the driving band, would thus

FIG. 19. FIG. 20.
FIG. 19.—ROLL. × 1. (PELOUX.)
FIG. 20.—THREE-INCH GUN AT ORIGIN OF RIFLING. × 2.25.± (TSCHERNOFF.)

be more preheated by the greater friction, and hence when exposed immediately afterward to the rush of the hot gases, would reach a higher temperature than the rest of the land.

The second piece of evidence is that, when a pressure plug is exposed in the powder chamber of the gun to 100 or 200 rounds, hardening occurs earlier in certain parts of certain edges of letters and other marks previously stamped upon it than elsewhere, and the earlier the greater the stamping pressure has been. This, too, is equivocal, because it

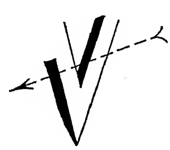

FIG. 21.—SUGGESTED DIRECTION OF GAS CURRENTS WHICH HARDENED THE PARTS OF PROF. FAY'S *A*, WHICH ARE BLACKENED HEAVILY IN THIS FIGURE. SKETCHED FROM HIS FIG. 6.

might well be that just those edges which hardened most were most favorably exposed to the hot gases, whether because they were sharper, or because they had a less broad path of escape for the heat into the cold body of the plug, or because they were on what was the lee instead of the weather side of the impression as sketched with the arrow in my Fig. 21.

That the pressure plug is exposed to severely eroding currents, in spite of its standing at the center of the rear face of the powder chamber

and thus out of the direct gas-path from breech to muzzle, is indicated by the erosion which has taken place in Prof. Fay's plug in the 100-round interval between the two stages shown in his Fig. 9, lessening the prominence of his letters stamped apparently under a pressure of 2000 lb. till it is less than that which those stamped with half that pressure had before this exposure.

Two sources of such currents suggest themselves, eddying, and convection due to the great temperature-difference between gas and metal. We incline toward the former because we are told that no erosion occurs in closed bombs, in which there should be convection currents much as in a gun chamber.

The erosion of the surface and the cracking of the rim of the pressure plug and of the circumference of its central hole suggests so strongly an order of magnitude of the erosion and cracking like that in the rifling, where the gas velocity is so enormous, that one who remembers how incomparably hotter a hot bath feels to a moving than to a still foot, suspects that the gases also to which this pressure plug was exposed must have been in violent motion.

Prof. Fay's micrographs suggest that the direction of the gas currents had much to do with determining the relative rapidity of hardening of the various parts of the letters stamped on his pressure plug. Thus in his Fig. 6, of which my Fig. 21 is a sketch, it is the right-hand edge of the male apex and the left-hand of the female apex of his letter *A* that is hardened. This effect might easily be caused by currents of hot gases passing from right to left, for these hardened edges would be the lee edges. But it does not fit the plastic deformation hypothesis readily, because it is not easy to understand why, if the deformation were stronger on the right than on the left of the male apex, it should be weaker on the right than on the left of the female apex.

Beyond this it is not self-evident that the variations in the deformation should have been such as to cause this peculiar distribution of the hardening at the apex of this letter *A*, and yet to cause the radically different distribution at the apex of the *V* in his Fig. 14. Again, how can local deviations in the deformation explain why the hardening tapers away sharply to nothing as we approach the female apex in my Fig. 21 on its left side? The severity of the deformation should increase as we approach the female apex, instead of thus dying out. On the other hand, if, as I suspect, the deviations in hardening represent deviations in heating, then the hardening might easily die out in this very way, because the lee-shore effect would decrease as the female apex is approached, and as the distance from lee to weather shore thus decreases.

And in general the difference between the right and left sides of Prof. Fay's stamped figures, while it could readily be caused by a difference in exposure such as would exist between the lee and the weather

sides of a cavity exposed to a violent stream of hot gases, is sharper than would be expected to result from an evidently unintentional variation in the deformation given the opposite sides by one as careful as he.

The distribution of the hardening in the cross-section through the driving edge of the land, as shown in Fig. 5, is equivocal. The greater thickness of the hardened layer at the salient $L$ is explained fully by the higher temperature which this part would reach because it is salient. Where in this section the greatest deformation would occur cannot be known with confidence. It might be at $L$, in which case the effect of the plastic deformation might simply reinforce that of the higher heating. Or it may be $K$, in which case Prof. Fay can justly explain that the excess of the temperature at $L$ so far outweighs the excess of plastic deformation at $K$ that the effect of the latter in increasing hardening is not detectable.

Prof. Fay's hypothesis receives some support from his observation that it was the obtuse and not the acute angle of the impression made by his period held at an angle of 45° that hardened the more.[22]

The obtuse angle would be expected to heat the less of the two. Hence its hardening the more may be due to some other cause, which might indeed be its greater deformation.

It is true that Prof. Fay[23] says positively that the hardening occurs earliest "where the *maximum* amount of cold work has been applied in stamping the letters." Unfortunately we cannot always if indeed often know beforehand where the maximum amount of cold work does occur. Hence I do not feel that we can accept this assertion without a thorough exposition of the evidence, and more particularly of the reasoning by which the parts of fastest hardening are identified with those of maximum plastic deformation.

In view of the equivocal nature of most of the evidence offered, it would seem well to experiment unequivocally by eliminating completely all effects of saliency, and hence to file away the irregularities caused by stamping letters on the surface of the pressure plug, leaving that surface perfectly plane. Then expose it to 200 rounds, etch it, and see whether these same letters re-appear, as they certainly will if plastic deformation hastens hardening.[24]

I understand that this suggestion of mine is to be carried out.

A salient must needs receive more heat, and hence must heat higher, and hence must needs harden the sooner and the deeper. That is not

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[22] *Trans.* (1916), **56**, 477 and 480, and private communication, Aug. 4, 1917.

[23] *Op. cit.*, 476.

[24] Frémont has shown by means of a most interesting experiment that the effects of plastic deformation extend well beyond the depth of impressions stamped on the face of the metal (*Bulletin de la Société d'Encouragement* (1903), Part 2, 362).

open to question. The effect of plastic deformation, on the other hand, is far from self-evident, and needs direct experimental verification. As long as any surface irregularities remain, there may be doubt as to whether the saliences suffice to explain the irregularities in hardening, and hence as to whether the evidence really proves that any part of this hardening is referable to the plastic deformation.

22. *In what Way may Plastic Deformation Hasten Hardening?*—Lt. Col. Dickson[25] explains reasonably that plastic deformation may hasten hardening by crushing the individual particles of ferrite and cementite to smaller size, by thus increasing the intimacy of their mixture, and by thus facilitating their merging to form austenite, a prerequisite to hardening, in accordance with the principles explained in Sec. 9.

The mere amorphization does not readily explain the hastening of hardening, because it is so rapidly effaced on heating. Metal initially made in part amorphous by plastic deformation reverts very quickly on heating to the condition which an undeformed piece would have at like temperature.

The increase in hardening caused by manganese and nickel, which lower $Ac_1$, of course points not to cold work but to temperature as the cause of hardening. It does not even most remotely suggest that prior deformation aids hardening.

The expression "the sorbite→troostite→martensite transformation" is to be taken figuratively, as meaning that, on quenching a series of specimens from an ascending series of temperatures, one gets first sorbite, then troostite, then martensite, because this is the order of the variation caused by such a rise of the quenching temperature, in the degree to which the austenite→martensite→troostite→sorbite→pearlite or *Ar* transformation is arrested by the quenching. The *Ac* transformation in heating is probably direct from sorbite or pearlite to austenite, without passing through the martensite stage.

23. *Possible Need of Coarsening.*—If it is true that plastic deformation hastens hardening by reducing the size of the individual particles of ferrite and cementite, and if hardening hastens erosion (Sec. 25), then hardening might be retarded and erosion possibly lessened by substituting some coarser structure for the sorbite now used for gun liners. This coarser structure would have a lower elastic limit, and hence this change would imply making the gun enough heavier to compensate for this. But if erosion is as serious a matter as we are sometimes told, the gain in life might be more important than this slight increase of weight. The experiments with steel coarsened somewhat by annealing from above $Ac_3$ are in line with this thought.[26]

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[25] Private communication, Aug. 10, 1917.

[26] Admiral Earle: *Trans.* (1916), **56**, 494.

24. *Carburization as a Cause of Hardening.*—Prof. Sauveur[27] brings up the case-hardening of the bore by the hot gases as a possible cause of bore-hardening. Their high temperature and pressure certainly would increase any carburizing action which they can have. But one asks first whether the gases contain a large enough ratio of carbonic oxide to carbonic acid and water vapor to give any rapid case-hardening effect, and second whether the time available suffices.

The carbonic acid content of the powder gases, as shown in Table 2, is always great enough to retard the carburizing action of the carbonic oxide materially, and in many cases, for instance Nos. 1, 5, 6, 7 and 9, to arrest it completely.

The time available, as we have seen in Sec. 3, is about 0.06 sec. per round for the breech of the gun, and about 0.02 sec. for my forward ring, or a total of 0.3 sec. and 0.1 sec. in the five rounds which have proved sufficient to cause appreciable hardening.

Even in the 170 rounds of my rings, the total length of time would be only 3.4 sec. for my forward ring. On its face this seems far too short a time to enable carbonic oxide, so much diluted with carbonic acid, to do any appreciable carburizing.

There is a natural further objection to the carburizing hypothesis, that at each round a layer 0.0005 in. thick is removed bodily by the erosion; that this is the layer which alone has been exposed to the powder gases and thus has been in a position to take up carbon; and that this time is manifestly too short to allow either the gases to diffuse into the metal or the carbon absorbed by the metal to migrate outward appreciably by the slow process of solid diffusion. This objection is weakened by the surf action indicated in Fig. 28, the molten layer rolling over and over, and both renewing the surface exposed to the gases, and distributing through the molten layer any carbon taken up at the bore surface. Even so, the time seems far too brief to enable carburizing to extend to the depth of the hardened layer of my forward ring, about 0.005 in. in addition to the 0.085 in. eroded in the 170 rounds (Sec. 1), or a total of nearly 0.1 in.

The inherent improbability of appreciable carburization is confirmed by Fig. 22, which shows the microstructure of the hardened layer and of the unhardened metal outside it, after re-heating to 1075° and cooling slowly. Note that the structure in the layer which had been hardened is now like that of the metal which had not, and in particular that the proportion of ferrite to pearlite is at least as great in the upper edge which had formed the hardened layer as in the region below it. This shows that no material carburization occurred.[28]

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[27] *Trans.* (1916), **56**, 484.

[28] This specimen, while still copper plated, was heated to and held at about 1075° in hydrogen for 2 hr., and was then cooled in the furnace, occupying 1 hr. 50 min. in cooling to 255°. It was then etched with picric acid.

Contributory microscopic evidence against the carburization hypothesis is that the martensite along the bore face is not more acicular than the rest, as it should be in case it were richer in carbon.

Confirmatory evidence is given the fact that the straight scratch made by a Turner sclerometer on the face of the specimen shown in Fig. 22 is not narrower at the bore face where the hardened layer had been than elsewhere, showing that this hardened layer is not richer in carbon than the rest.

Incidentally, the grouping of the ferrite about the crack shown in Fig. 22 and about all the other cracks in this specimen is an additional proof of my contention that that which causes ferrite to assemble about

FIG. 22.—REAR RING, LONGITUDINAL SECTION. × 79.

slag bodies is not some obscure attraction which the slag has for ferrite, but simply the expulsive power of the austenite grain, which causes it, like every other crystalline body, to try to expel foreign matter to its outside.[29] Slag bodies lie outside the austenite grains. During the transformation the austenite later expels to its own outside the pro-eutectoid ferrite as fast as it forms, thus depositing it about the already present slag bodies.

In the present case, we cannot conceive that a mere crack has any possible attraction for ferrite or anything else. Hence the ferrite now lies about the crack simply because this is part of the outer surface of the austenite grains which, before the transformation, abutted against it. They expelled the ferrite hither during the transformation.

[29] See the Author: *The Metallography of Steel and Cast Iron*, 170–1 and 280–4, McGraw-Hill Book Co., Inc., 1916.

25. *Does Hardening Increase Erosion?*—There is reason to suspect that it does, especially during period *AB* of Fig. 2. The extreme brittleness of hardened steel gives it a tendency to crack in changing temperature abruptly, because of the consequent abrupt variations in volume from layer to layer. Once the temperature rises high enough to anneal the steel, that is, to replace the brittleness of martensite with the ductility of sorbite, the changes in temperature cease to cause cracking. In carbon steel this change under usual conditions of heating appears to complete itself at about 400° C., but under the extremely rapid heating in the gun tube the martensite state may well persist, with its attendant brittleness, to a materially higher temperature. To avoid debatable ground, let us confine our attention to temperatures between the atmospheric and 400° C.

In period *AB* the bore face is exposed to the hot gases under enormous pressure at about 3000° C., and hence heats very rapidly. Its heating to 400° C. may well outrun that of the layer next outside it, the subcutaneous one, so rapidly that the tangential compressive stress thrown upon it may be great enough to cause it to buckle and thus to flake away from that layer.

Turning now to the hardening period, *JM*, in which the successive layerets of the layer-to-be-hardened are hardening one by one, as each of them changes from austenite to martensite it undergoes a sharp expansion which, passing as a wave, exaggerates the tendency to crack. This wave in a bar quenched in water passes from its surface into its body, and the expansion of the subcutaneous layer tends to stretch the rapidly cooling hardened brittle surface layer beyond its limit of elongation, and thus to crack it. The wave of hardening in the gun liner passes in the opposite direction, from the body of the metal toward its surface, the face of the bore. Here the expansion of the subcutaneous layer in passing $Ar_1$ is thrown upon the bore face while it is still above $Ar_1$ and therefore unhardened, therefore still ductile, and therefore not susceptible to flaking off.

On this account such flaking off as is caused by the creation of a hardened layer seems likely to occur during period *AB*, that is, during the heating up, and not during the cooling down.

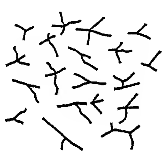
If such flakings were large, they should leave easily recognized traces in the section of the gun liner, because, occurring in the beginning of the heating up, they would cause a corresponding undulation in the outer boundary of the hardened layer. The extreme regularity of this boundary, both in longitudinal and in transverse section, and the absence of all indication of such flaking to the naked eye in all parts of the gun ring, argue that large flakings are rare if they occur at all. The suggestions of flaking at *B* and *C* in Fig. 12 and at *B* in Fig. 13 and 14 probably represent injuries done in preparing these microsections, because there is no corresponding depression of the outer boundary of the hardened

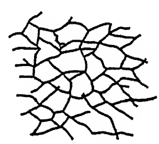
layer. Here let us remember that if flaking occurs it should be at the beginning of the heating period, say in the early part of Period *AB* of Fig. 2, thus leaving ample time for the outer limit of the hardened layer to deflect correspondingly.

These speculations must of course be received cautiously.

## II. The Cracking of the Bore

26. *Appearance of the Bore and Cracks in Elevation.*[30]—On repeated firing, the surface of the bore becomes covered with fine cracks. According to Tschernoff, at the origin of rifling the cracks are like those in Fig. 20, recalling the drying cracks in clay, the hardening cracks in steel, and very exactly the cracks on the surface of a roll, Fig. 19, of which each element of the surface, after becoming heated by contact with hot metal, is next chilled rapidly by the cooling water of the roll. Where there is a strong dynamic effect the most prominent and continuous cracks are transverse on the lands, but longitudinal in the grooves, following the direction of the rush of gases, as shown in Fig. 16.





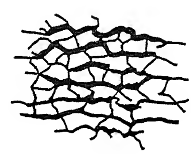


Fig. 23. Fig. 24. Fig. 25.

Fig. 23, 24, 25.—Tschernoff's sketches showing formation and development of cracks in gun liners.

The cracks begin as little three- or four-rayed stars, Fig. 23, the rays of which gradually extend till they form a network as in Fig. 24. On the centering cone and at the origin of the rifling, the cracks which happen to run longitudinally of the gun lengthen and widen faster than the rest, thus leading, we may infer, to the strongly marked longitudinal systems of cracks, Fig. 17 and 25.

27. *The Cracks in Elevation in my Rear Ring.*—The whole inner surface of my rear ring is bright, of the color of ground but unpolished steel, save that the cracks are black. In each system of 1 land + 1 groove, the chief crack usually runs longitudinally at the bottom of the driving edge of the land, as above land 4 and below groove 7 in Fig. 17. These cracks are often almost perfectly straight for the whole of the $1\frac{5}{16}$-in. width of the ring, especially where the remains of the rifling are still

[30] The assertions in this paragraph and the following one are due to Prof. Tschernoff (*Journal of the U. S. Artillery* (1914), **41**, 346), excepting those referring to Fig. 19, which are due to Capt. H. Peloux (*Idem*, Fig. 2, 346). The rest of the description applies to my two rings.

prominent, but in a considerable proportion of cases, especially where the rifling has been almost completely effaced, they are displaced by stepping, as at grooves 6 and 7, in Fig. 17.

These longitudinal cracks naturally tend to form at the base of the driving edge of the land, for at this re-entrant angle the change of section is sharpest. This tendency naturally becomes weaker as the effacement of the lands becomes more nearly complete.

In addition to the irregular longitudinal cracks, nearly all of which are at least roughly parallel to the rifling, there are distinct fine very straight scorings, which instead are parallel to the axis of the gun. Some of these are prominent on land 5 of Fig. 17. Their parallelism to the axis shows that they are formed by the gases and not by the driving band, the forward motion of which is spiral, though it may be that here, at the origin of rifling, and after the erosion has proceeded so far, the projectile may not be engaged effectively by the now nearly obliterated rifling, and hence may move forward straight instead of spirally.

Moreover, their straightness and continuity contrast with the irregularity and stepping of the main spiral cracks quite in the way in which such true scratches made by the forward moving gases or projectile should contrast with the contraction cracks formed in cooling after the projectile has left the gun, in period *FN* of Fig. 2. I find no clear indication of copper in the rear ring.

28. *The Cracks in Elevation in my Forward Ring, Fig.* 16.—Here the face of the lands and a little of the strongly concave surface to right and left, about from *a* to *a′* in Fig. 1, is of a bright iron color, as if ground but not polished, and so is a narrow line at *b* in Fig. 1 and 16. But the grooves are black, with here and there a network, its main lines running longitudinally. There is a distinct copper color in parts of a good many of the grooves, suggesting that they have been rubbed in handling since the ring was cut from the gun.

29. *The Copper Network in the Grooves.*—In the grooves the cracks form a network corresponding roughly to that of the more prominent cracks in Fig. 20, while the meshes correspond roughly to the cracked spaces within that network. The network is shown in groove 1 of Fig. 16.

On heating a fragment of the forward ring in hydrogen, most and perhaps all of the cracks are found to be filled with copper as shown rather obscurely in Fig. 11. A careful examination of the longitudinal and transverse sections indicates that this copper reaches nearly, if not quite, to the roots of the cracks, but in many cases it does not reach to their tops, that is it does not fill them flush, though from many cracks of the main network it projects in the form of a thin network of dykes.

In the elevation of the ring as received this copper is masked, be it by oxidation, be it by ash or carbon, so that the copper-red of the dykes

can be recognized only here and there, and then usually only with a lens. But on rubbing lightly with either emery paper or unglazed porcelain the network is developed, so that its prominent dykes, surrounding meshes which are still black, can be seen even with the naked eye, as in groove 3 of Fig. 16. In this rubbing the porcelain receives a gray streak, which dissolves away completely in nitric acid, as if it consisted of cupric oxide rubbed away from the summits of the copper dykes and from the black meshes between them.

The copper network is seen on further rubbing to fill most of the important longitudinal cracks, and many of the transverse ones, and at this stage of the rubbing to fill them flush to their mouths. The present condition is shown in groove 2 of Fig. 16. On a larger scale it is much like that shown in Fig. 18, in which the white masses are the initially black meshes now rubbed bright iron gray, the light gray network is the copper which now fills certain of the cracks full, and the black network is the remaining cracks in which no copper is yet seen. Those parts of the meshes in which the steel does not rise quite to the general level, seem often to have been smeared over with copper by the rubbing.

On now heating in hydrogen, most, if not all, of the cracks which hitherto have remained black become copper-colored.

The appearance of the grooves after the complete removal of the copper is shown in the right hand part of groove 2 of Fig. 16. Here the copper has been removed by brief exposure to strong nitric acid, which throws the iron into the passive state.

This copper filling of the cracks is not seen clearly in my photographs, Fig. 5 to 10 and 12 to 15, primarily because in grinding and polishing the section for microscopic examination the soft copper becomes grooved so deeply and concavely that it reflects but little light into the microscope. Suggestions of this filling are seen in the left-hand cracks of Fig. 7 and 12, and in the right-hand one of Fig. 14.

30. *The cracking of the lands*, Fig. 16 and 18, differs very prominently from that of the grooves in that the main cracks are transverse, instead of being longitudinal. There are a few long continuous and very straight longitudinal cracks in land 3 of Fig. 16. These, with the main transverse cracks, divide the face of the lands into rectangular meshes, most of which in turn are broken up into roughly rectangular patches by minor longitudinal and transverse cracks. These little patches in turn are roughened with little furrows as if they had been ground or filed longitudinally with a fine file. This roughening cannot be seen with the naked eye. It is prominent in Fig. 18.

Further differences are that the lands are not black, but iron gray like the rear ring; that copper does not protrude from any of their cracks; and that the proportion of cracks here which on mere rubbing show copper is much smaller than in the grooves. These copper-showing

cracks are filled flush with copper, while the rest are black and not filled flush. Yet these black cracks contain copper. They seem to owe their blackness to an oxide coating, because after heating in hydrogen the bright copper color is seen in all of them, except those at the sharpest part of the salience, *a* of Fig. 1.

Thus here as in the grooves the cracks may be divided into the main ones, forming a major network, rectangular in this case, and the minor ones forming a nearly rectangular network of a lower order of magnitude; and they may be cross classified into those showing copper initially, all of which are filled flush, and the initially black ones, which contain copper but are not filled flush with it to their mouths.

31. *Appearance of the Cracks in Section.*—In order to make their appearance clear, I had both transverse and longitudinal sections of my forward ring copper plated so as to support the thin hardened layer in grinding and polishing.

In both longitudinal and transverse sections the hardened layer is seen as a thin light band, about 3 mm. (1/8 in.) thick, bordered with a thick black band above and a thin one below, the former representing the surface at the junction of the harder steel and the softer copper, which becomes inclined in grinding and polishing the microsection, and hence reflects no light into the microscope; the latter representing not only a like inclined surface at the junction of the hardened layer and the softer unhardened metal, but also probably a band of troostite, formed as described near the end of Sec. 17.

In both sections the cracks are spindling slices, carrot-shaped in section. That they hold in depth as slices and not as carrots, I proved by taking three photographs of the same part of a longitudinal section. Fig. 8 shows the initial condition, Fig. 9 and 10 the condition after grinding away 0.5 mm. and 1 mm. Note that the main cracks, *A*, *B*, and *E*, coincide in position and tolerably well in depth in the three photographs.

The transverse cracks in the lands and the longitudinal ones in the grooves are very much alike in section. Though their sides are often straight as seen in section, yet they are curved or jagged in a considerable proportion of cases. Moreover, the opposite sides of a crack, though they usually match each other pretty well, yet do not always, as, for instance, in Fig. 10. This must be taken rather cautiously, because in grinding and polishing the section the sides of the cracks may become flowed out of true.

The sharpness of the bottoms of the cracks, for instance, in Fig. 5, 6, 8, 10 and 13, should be noted carefully. The spindling of the cracks shows that they are torn open mechanically. Cracks deepened by abrasion would be round-bottomed.

The cracks have no tendency to stop at the outer edge of the hardened

layer. Thus of the seven cracks in Fig. 5, three end in the hardened layer, three pass beyond it, and only one stops at its outer face.

Note that the depth of the deepest cracks is about 0.9 mm. (0.035 in.) or about 7.5 times the thickness of the hardened layer, 0.0047 in., the bore face of which reaches about 1250° at the instant of highest heating, while its outer layer limit reaches about 750°. We have thus in passing outward through this first 0.0047 in. a fall of about 500° in the maximum temperature reached. In view of the steepness of this thermal gradient at this time, the highest temperature reached at the bottom of the cracks, or 7.5 times as far from the bore face, certainly ought to be very low. It is hard to believe that the metal here ever became so hot that its contraction on cooling would, if unaided, be enough to cause a crack to continue. Therefore this great depth, 0.9 mm., seems referable rather to some mechanical condition, than directly to the expansion and contraction of the metal here.

32. *Appearance of the Hardened Layer.*—There are four striking features of this layer: first, that it is of nearly the same thickness throughout the forward ring, except for being thickened on the driving edge of the land and for having fragments torn out of its inner or bore face; second, that its outer face, next to the unhardened steel, is very nearly straight, and unaffected by the local irregularities of its bore face; third, that it has very little if any tendency to follow down the sides of even the widest cracks; and fourth, that there is no sign of a cornice such as would be expected in the longitudinal section from the forward scraping action of the erosion on the very face of the bore, especially on the lands.

That its thickness is the same on the groove and the land is seen on comparing Fig. 7 and 15. Its thickening on the driving edge of the land is shown in Fig. 5. How little its outer face is affected by irregularities in its bore face is shown in Fig. 12 and 13.

Its failure to follow down the sides of the cracks is shown in Fig. 5, 6 and 13. As near an approach to following down the crack as any that I find is shown at the left-hand edge of the left-hand crack in Fig. 7.

The complete absence of any suggestion of a cornice even in the longitudinal sections is illustrated by Fig. 7, 12, 13 and 15.

Let us now seek the explanation of these phenomena.

33. *The longitudinal cracking in the grooves* would naturally be expected. Of the first little checks in the powder chamber and in the grooves, running indiscriminately in all directions, Fig. 24, those which run lengthwise of the gun are naturally scored and widened by the abrasive action of the gases, Fig. 25. The deepest and most marked of these scorings would naturally form the starting points of the contraction cracks which develop.

34. *The Absence of Hardening in the Cracks.*—The failure of the hardening to follow down the sides of even the widest cracks harmonizes

with the sharpness of the crack-ends in arguing that the gases are not active in the cracks themselves. If they entered into a crack to a depth corresponding to their enormous pressure, they ought certainly to cause the hardened layer to follow down into it, though, of course, taperingly. That it does not argues that the cracks are obstructed (Sec. 37).

The narrow light band on the left-hand side of several of the cracks might at first be taken for a continuation of the hardened layer. It is shown to be an optical illusion by rotating the specimen under the microscope. Indeed, this might have been inferred from its being always on the left-hand side of a crack.

35. *The Transverse Cracks on the Face of the Land.*—Tschernoff's explanation[31] of the transverse cracks on the face of the lands is that the surface of the land, *ABCD*, Fig. 26, is prevented from expanding longitudinally during its rapid heating by its integral union with the still

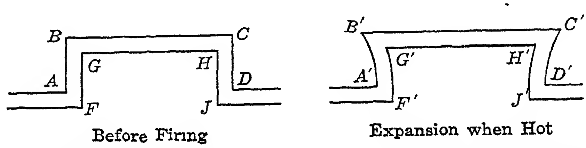



FIG. 26.—IMAGINARY CROSS-SECTION THROUGH A LAND BEFORE AND DURING EXPANSION.

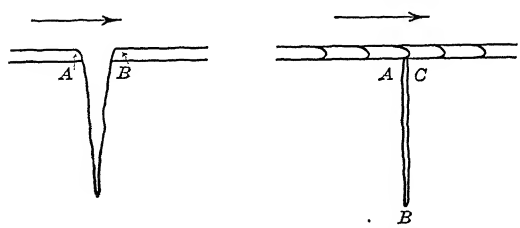



FIG. 27. FIG. 28.

FIG. 27.—A CRACK WIDE OPEN AFTER COOLING.
FIG. 28.—THE SAME CRACK CLOSED BY EXPANSION AND WITH THE SOFTENED METAL FLOWED PAST IT.

cool outer parts of the land, *FGHJ*, and their union with the main body of the liner, neither of which expands longitudinally. But the upper part of the land is free to expand laterally, as at *B′C′*. Thus while the wave of heat is moving outward through the land, the heated parts of the land are in the condition of a spring, in that their tendency to expand longitudinally is restrained, and that their transverse expansion is increased correspondingly, because their total volume no doubt increases in almost exact accordance with their rise of temperature. In this deformation the parts of the land enclosed within the lines *FGHJ*, in taking on the shape *F′G′H′J′*, may be assumed to remain within their

[31] *Journal of the U. S. Artillery* (1914), **41**, 346.

elastic limit, so that, in again cooling, they tend to return exactly to their initial shape *FGHJ*, that is, to contract chiefly transversely and but little, if at all, longitudinally. But the higher temperature reached by the layer between *FGHJ* and *F′G′H′J′* probably carries it far beyond its elastic limit, because with that higher temperature go both greater expansion and a much lower elastic limit.

If, to simplify our ideas, we assume for the moment that the elastic limit of this layer when it is hottest is nil, so that it is under no longitudinal stress, then in cooling it would endeavor to contract equally in all directions, longitudinally as well as transversely. Because it is integrally united with the inner part, *F′G′H′J′*, which is not contracting longitudinally at all, its effort to contract longitudinally will throw it into correspondingly great longitudinal tensile stress, not only beyond its tensile elastic limit, but even beyond its power of tensile elongation, so that it cracks transversely.

36. *Need of an Additional Explanation.*—Though this explanation is valid qualitatively, it hardly explains either why these transverse cracks occupy so great a proportion of the hardened layer, or why they are so much greater on the forward lands than in the forward grooves or in either groove or land of the rear ring. The freedom of the forward lands to expand transversely explains the slightness of their longitudinal cracks, but it does not explain the excess of their transverse cracks over those of these other parts. To this I will return in Sec. 42 and 43.

In a series of photographs which collectively represent a running length of 1³⁄₃₂ in. (27.5 mm.) of the longitudinal section of my forward ring, I find that 16.9 per cent. of the length of the section is occupied by cracks which pass materially beyond the thickness of the hardened layer. Fig. 12, 13 and 14 are selected from this series. The width of these cracks is measured, not at their very mouths, but at the side of the hardened layer next to the unhardened metal, and, therefore, where the temperature probably rose at most but moderately above $Ac_3$. It is true that the apparent width of the cracks is magnified considerably by the rounding off of their edges in polishing the microsection. But this is hardly more than enough to balance our measuring the width at the outside of the hardened layer instead of at the bore face. Indeed, on the bore face itself, as shown without any polishing in Fig. 18 and 20, the proportion of the surface occupied by cracks is of this order of magnitude. If, to be generous, we were to admit that the temperature rose here to 935°, or about the finishing temperature of rails at the hot saw, we should be confronted with the fact that the contraction in cooling from this temperature, as provided for in the rail specification of the American Society for Testing Materials, is only 6.75 in. (170 mm.)[32] in

[32] *Year Book of the American Society for Testing Materials* (1916), 24.

33 ft., about 1.7 per cent., or only one-tenth of the proportion represented by the transverse cracks in these lands. Even in cooling from the solidus down the contraction provided for by steel founders is only ¼ in. per foot, or 2.1 per cent. It is true that the true contraction from the solidus is somewhat greater than this, because the lag of the slower cooling interior stretches the faster cooling exterior beyond its normal dimensions, and this in turn, once grown rigid, resists the lagging contraction of the interior. But even in a test in which the effect of æoliotachic contraction was reduced by casting 1-in. square bars 1 ft. long between the faces of a yoke machined to size, the contraction of carbon-steel castings was only 0.2875 in. per foot, or 2.4 per cent.[33]

Clearly, then, while the Tschernoff explanation, the unbalanced longitudinal contraction of the lands, is satisfying qualitatively, it is not quantitatively, failing to explain why so much of the length of the land is occupied by transverse cracks.

37. *Is the Cracking Cumulative?*—Our natural thought is that the cracks become widened cumulatively by foreign matter which, entering them when they have been left open by the contraction which follows the expansion of heating, prevents their sides from returning during the next heating to the exact contact with each other which the metal now forming their sides had during the initial expansion when there were no cracks. The metallurgist knows that a combination of strong buckstaves and tie-rods is needed to prevent his furnaces from cracking open in cooling down, and receiving then into their open cracks foreign matter, such as that crumbled from the crack-sides, which prevents the cracks from closing completely during the next heating. If the thickness of such foreign matter is called $t$, then because in the second contraction the walls contract exactly as much as they did in the first, the sum of the width of the cracks at the end of the second cooling is greater by $t$ than that at the end of the first cooling. The entry of a new lot of dirt, of thickness $t$, during the second cooling causes the sum of the cracks after the third cooling to exceed that after the first by $2t$, and so on till the furnace walls are so far out of shape as to be useless.

The cracks which form in a gun tube in the contraction period, *FN* of Fig. 2, may be prevented from closing completely during the expansion period, *OK*, of the next round by the entry of copper bitten from the driving band, or of carbon or ash from the gases, or by the formation of oxide. Let us consider these.

38. *The Copper-mouthful Hypothesis.*—The cumulative widening and deepening of a transverse crack on the face of a land by means of copper thus bitten off is sketched in Fig. 29, the first stage of which shows a newly formed contraction-check on the face of the bore. The lips of

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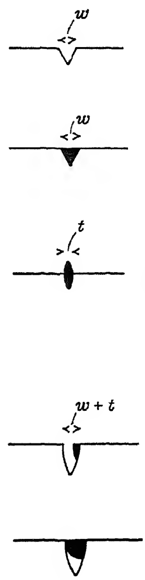
[33] J. H. Hall: Private communication, Aug. 1, 1917.

this check are still cool during the period $OA$, Fig. 2, of the next round when the driving band is scraping across them, and because cool they still retain their hardness received in period $MNP$ of the rapid cooling of the preceding round, and because they are thus hard and cool they are in an excellent condition to bite a mouthful of the soft copper off the passing driving band, as sketched in Stage 2.

During the heating and expansion which now follow, the sides of the crack, in attempting to close, probably squeeze part of the copper thus bitten off outward toward the root of the crack, and part of it inward,




Stage 1 — A crack starts during the cooling of a given round. Its width is $w$.

" 2 — During Period $OA$ of the next round (Fig. 2) the hardened steel lips of this crack bite a copper mouthful from the passing driving band.

" 3 — During the remainder of the heating and expansion of this round the closing of the crack extrudes part of the copper mouthful. The copper not thus extruded prevents the complete closing of the crack and thus shortens the section between cracks by the amount, $t$.

" 4 — The shortening of this section persists during the cooling of this round, thus widening the cracks by nearly $t$ and thus deepening them.

" 5 — In Period $OA$ of the next round the crack thus widened bites off enough more copper from the driving band to fill its widened mouth.

FIG. 29.—SUCCESSIVE STAGES IN THE WIDENING OF A TRANSVERSE CRACK IN THE FACE OF A LAND, BY BITING OFF COPPER FROM THE DRIVING BAND.

extruding it into the bore, as sketched in Stage 3. The copper thus extruded is wiped away forward by the rush of gases, though, as we have seen, this rush is not strong enough in the grooves to wipe away fully that which is extruded, so that some of it remains as a dyke-network.

So much of the copper as is not extruded but remains in any given transverse crack prevents that crack from closing completely, and thus prevents the longitudinal expansion of period $AG$, Fig. 2, from completing itself, and thereby causes a corresponding fraction of this expansion to take place radially and circumferentially as in Fig. 26. This is the important conception, this effective shortening and widening of the section of the land between adjoining transverse cracks. Because this section is thus effectively shortened, it starts in the contraction of the cooling of this round correspondingly shortened, and undergoing the same pro-

portion of contraction during this cooling arrives correspondingly shortened at the end of the contraction, and hence the transverse cracks which bound it are correspondingly widened as in Stage 4 of Fig. 29. Of course that which thus increases from round to round is not the actual open space of the cracks after cooling, which should remain constant and represent the true longitudinal contraction due to the cooling, but the distance from steel wall to steel wall at the various cracks, which represents the sum of this contraction plus the copper and any other filling which keeps the steel walls apart. It is this sum that constitutes the cracks as we see them in Fig. 5 to 14. This widening of these cracks naturally tears their bottoms deeper, that is farther outward from the bore.

In period *OA* of the next round, Fig. 2, it is the now widened crack which bites off copper from the passing driving band, as in Stage 5 of Fig. 29.

Here, then, we have a cumulative process, each crack stopping its own mouth with copper bitten from the driving band, and being prevented by that copper from closing completely in the immediately following expansion, thus starting each cooling and contraction wider by just so much than at the corresponding stage of the preceding cycle, and thus growing in width round after round.

In Sec. 42 we shall see that oxide and perhaps carbon may add to this exaggeration of the width of the cracks.

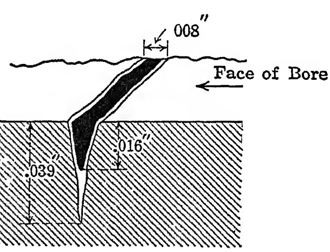



FIG. 30.—SLIVER OF COPPER, AND TRANSVERSE CRACK IN LAND OF FORWARD RING IN WHICH IT LAY.

39. *Evidence Supporting this Hypothesis.*—It was after an analysis of the conditions had suggested this copper-mouthful hypothesis that by the careful study of these cracks and the reduction of the copper oxide by heating in hydrogen, outlined in Sec. 29, I detected the presence of the suspected copper. The actual presence of slivers of copper in the cracks was further proved by trepanning a thin slice of the metal parallel to the bore face of the forward land, annealing it in hydrogen, and opening certain of the cracks by bending it with the bore-face convex. This partly detached the copper fillings, one of which now stretched diagonally across its crack, so that it could be examined well. On slight further

bending it fell out and proved to be a triangular prism of copper, with one side coated with a dark substance. This copper sliver and the crack in which it lay are sketched in Fig. 30. Note that the crack itself reaches some 0.023 in. below the copper stopping.

Both walls of this crack were now of a reddish-brown copper color, caused by a very thin coating of powder. Scraping this off lightly disclosed the normal iron gray behind.

The fact that, though the copper seems to fill the various cracks to their roots, yet this and other slivers of copper which I have removed by like means are too short to reach to these roots, shows that the copper in each crack is not welded firmly into a coherent whole, but breaks up on bending the metal. The pieces which make up these crackfuls are probably so small that only the largest ones are found on removing them thus by bending.

40. *The Alloying of Copper.*—Where a copper crack-filling is flush with the bore face or protrudes beyond it at the end of a given round, in the heating period of the next round it should be heated hot enough to alloy with the iron. At the very hottest layer, that is at the very face of the bore, this alloying should be most rapid, and the resultant alloy would naturally form an important fraction of the metal entrained as spray. Yet the hottest part of the metal not sprayed away, but scraped forward as in Fig. 28, may well hold an appreciable quantity of copper.[34] The small carbon content, 0.38 per cent., would not prevent this. In the extremely rapid cooling the expulsion of the copper due to the rapid decrease of its solubility with falling temperature may not have time to cause it to collect in detectable drops. But several pieces which I cut from these rings and annealed in hydrogen, with furnace-cooling down to dull redness, were coated abundantly with protruding drops of copper. Even a strip cut from the rear ring was thus coated, though much more sparingly than those from the forward ring. In one case these drops occurred as much as 0.1 in. (2.5 mm.) from the bore face, that is very far outside the hardened layer, and even beyond the roots of the cracks. As the copper could not have reached here in the gun by diffusion, in view of the relatively low temperature, say 750° reached at the outside of the hardened layer and in the few seconds total time available (Sec. 3), we may infer that it dissolved and diffused outward in my annealing, to re-segregate in the form of drops in my slow cooling.[35] Hence the absence of local segregates of copper in the gun rings as received.

[34] J. E. Stead: *Journal of the Iron and Steel Institute* (No. 2, 1901), **60**, 118; R. Sahmen: *Zeitschrift für anorganische Chemie* (1908), **57**, 9; Guertler: *Metallographie* (1912), **1**, 86.

[35] Annealing a second lot of specimens at 950–960° in hydrogen for 75 min. with faster cooling did not develop these copper drops, probably because the opportunities for migration were less.

41. *The Absence of a Cornice.*—The conception which we have just formed, that the cracks stop their own mouths with copper which they bite from the driving band, must be carried even farther to meet the remarkable absence of any sign of a cornice from the edges of the transverse cracks. In the longitudinal section, Fig. 7, I find no indication as to which is the muzzle and which the rear end. This means that the cracks remain closed during the whole erosion period, for if they were opened, then the softened metal which is dragged forward by the rush of gases would form a cornice on the weather lip of each crack, *A*, Fig. 27, and would round off the lee lip *B*. The absence of all indications of forward and aft in this section seems to imply that the mouth of the crack is not only closed but welded across during the whole period when erosion is rapid, perhaps somewhat as sketched in Fig. 28. We can imagine that the steel which is melted and that heated erodibly hot, together with the extruded copper, are swept forward as a pasty layer, any given molecule perhaps flowing far forward of its initial position. It should be this forward flowing that causes the choking of the bore toward the muzzle.

When the period of contraction comes, at *K* in Fig. 2, it is the pasty and as yet unhardened layer on the very face of the bore that first begins contracting and tending to crack. It is only an extremely thin layer that has thus been pasty enough to weld, and the rest of the walls *ABC* of the crack, Fig. 28, are still unwelded. The new crack which now forms in the pasty layer would have a certain tendency to locate itself so as to form a new mouth for the unwelded crack-root which is thus already present, reaching almost to the surface, thus really re-constituting the same old crack, and enabling it to take a second bite at the driving band, as in Stage 4 of Fig. 29. Yet the tendency of this pasty inner layer, in starting to contract, to open exactly opposite an old crack would be by no means completely dominating, and hence many new cracks may form in the place of old ones of long standing. When the wave of contraction later passes from the pasty layer to the deeper-seated ones it will naturally tend to re-open old existing cracks. Each of these, in re-opening, would naturally tear through the pasty layer and thus re-open itself to the bore. It may be thus that the great number and greatly varying depth of the cracks arise.

This forward sweeping of the pasty layer probably continues into the period of cooling and contraction, and prevents the very first part of the contraction from actually causing cracks, because any which started to open would at once be flowed over by the forward moving layer of pasty metal. The absence of a cornice indicates that the cracks do not open till the bore-face pasty metal has grown so cool and rigid that it is no longer capable of being flowed forward by the now enfeebled stream of gases.

42. *Further Exaggeration of the Width of the Cracks by Oxidation.*—The composition of the powder gases, according to several analyses kindly obtained for me by Mr. Hudson Maxim from Captain A. T. Kearney, Acting Chief of the Bureau of Ordnance, United States Navy, lies within the limits shown in Table 2.

TABLE 2.—*Composition of Gases from Combustion of Powder*

| No | Description | Composition, Volume, Per Cent. | | | | | | | Ratio $CO_2:CO$ |
|---|---|---|---|---|---|---|---|---|---|
| | | $CO_2$ | CO | H | $CH_4$ | N | O | $H_2O$ | |
| 1 | Nitrocellulose fired in bomb *in vacuo*, dry gas | 21 | 42 | 19.7 | | 17 | | | 1:2 |
| 2 | Gas from 5-in. 50-caliber gun, Navy smokeless powder, dry gas | 13.8 | 40.1 | 7.14 | | 35.54 | 1.00 | 2.10 | 1:2.9 |
| 3 | Gas from 3-in. 50-caliber gun, Navy smokeless powder, dry gas | 6.10 | 40.8 | 5.76 | | 45.14 | 1.9 | 0.30 | 1:6.7 |
| 4 | From Russian publication, same type of powder, dry gas | 15.9 | 47.7 | 22.6 | 1.4 | 12.4 | | | 1:3 |
| | Experiments of Noble and Abel, 1868–1874 | | | | | | | | |
| 5 | Four English powders | 26 | 3 | | | 11 | | | 1:0.12 |
| 6 | Spanish powder | 25 | 1 | | | 11 | | | 1:0.04 |
| 7 | Blasting powder | 23 | 15 | | | 9 | | | 1:0.65 |
| 8 | Nitrocellulose (gun cotton) $C_6H_7O_2(NO_3)_3$, theoretical gas on assumption that all the H is oxidized | | | | | | | | 1:3 |
| 9 | Nitrocellulose + 25 per cent. nitroglycerine on the same assumptions | | | | | | | | 1:1.6 |

1 to 4, Capt. T. A. Kearney, Acting Chief of Ordnance, U. S. Navy, Aug. 27, 1917, by the kindness of Mr. Hudson Maxim.

5, 6, 7, Capt. Henry Metcalfe: *Ordnance and Gunnery*, Chap. IX, p. 6, Wiley, N. Y., 1891.

The sum of the carbonic oxide and hydrogen is so large and the time available so brief (Sec. 24) as to give the impression that oxidation is hardly to be looked for. The ferromagnetism of most of the little slivers which I detached from the cracks may be due to little particles of the hardened layer chipped off with them. Moreover, the bore faces of the rings are metallic, and, rough as they are, do not rust on exposure to the air as they might be expected to if initially oxidized. Hence the red powder found on the walls of the crack may not be oxide but the ash of the powder.

Yet oxidation might occur. Bell[36] found that even pure carbonic oxide oxidizes metallic iron while depositing carbon, at or below 417°. Baur and Glaessner[37] found that at 870° and 510° iron oxidizes to ferrous oxide even when the ratio of carbonic acid to oxide is about 1 : 2, and that the proportion of carbonic acid to oxide needed to cause oxidation de-

[36] *Journal of the Iron and Steel Institute* (No. 1, 1871), 163–4.

[37] Findlay: *The Phase Rule* (1906), 308; *Zeitschrift für physicalische Chemie* (1903), **43**, 354.

creases rapidly as the temperature rises above 870°. Bell's observation suggests a like decrease as the temperature sinks below 510°. Moreover, though the total time available for oxidation at temperatures above 870° must be extremely brief, that available below 510° should be much longer. Iron oxidizes in air at temperatures at least as low as 149°.[38]

Because iron expands greatly in oxidizing, any oxide formed on the sides of the cracks would act like so much foreign matter, and coöperating with any ash and carbon from the powder would reinforce the action of the copper mouthful. Moreover, their action might well reach to the bottom of the cracks, thus explaining the presence of the reddish powder (Sect. 39).

43. *Are the Prominent Transverse Cracks Chatter-Marks?*[39]—Having now seen how the width of the cracks in general may be exaggerated, let us next ask whether the excess of the transverse cracks on the forward lands over those on the forward grooves and on both lands and grooves in the rear ring is not due to their being exaggerated by chattering.

This excess is seen prominently on comparing Fig. 16 and 17, and also in Prof. Fay's Fig. 1, and even better in Photo 1, *Tests of Metals*, 1913, facing page 134.

I suspect that they are chatter-marks, first because they are confined to those surfaces against which the driving band presses severely, and second because of their rough approach to periodicity. Chatter-marks indeed represent vibration, but this vibration is readily caused by severe pressure between a moving object and a stationary one, as when a lathe-tool takes a cut heavier than befits the rigidity of the lathe, or when a glacier cuts with its rock plowshare deeply into the ledge, or when a contractor's loaded scraper is drawn over clayey ground. This severe pressure clearly exists on the lands forward, but not in the grooves, nor on the rear part of the lands which have already been so far eroded that the driving band simply rests on the bottom ones and does not reach the top ones. We may suppose that these rear parts of the lands became thus deeply chatter-marked during the early rounds, when they still received severe pressure from the driving band, but that these marks have in turn been effaced by the erosion.

The position of these chatter-marks would naturally be such that they would coincide with the deep cracks, the roots of which remain unwelded from previous rounds (Sec. 31). Thus arises the further exaggeration of certain prominent transverse cracks.

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[38] J. E. Howard (*Report of Chief of Ordnance, U. S. Army* (1893), 703), found that steel oxidizes to a straw color at 149° C.

[39] "Chatter-marks, irregular tool-markings caused by vibration due to lack of stiffness in a lathe or similar machine. Transverse crescentic marks in a continuous series, sometimes occurring in deeply gouged glacial striæ as the result of vibration." *Standard Dictionary* (1898), 323.

44. *The Greater Prominence of the Longitudinal than of the Transverse Cracks in the Rear Rings* is shown in Fig. 17. I find no better explanation of this than that it is the concentration of the longitudinal cracks into a smaller number, each of them wide, that gives this prominence, and that this concentration is due to the abruptness of the variations in the transverse section. Notice how many more transverse than longitudinal cracks there are.

We naturally remember here that the riveting of the longitudinal seams of a boiler is doubled, because the ratio of transverse stress to longitudinal section is double that of longitudinal stress to transverse section, a principle which in the case of a gun is reinforced by the weakening of the longitudinal stress by the kick of the gun and the forward travel of the projectile.

But how is this undoubted cause to lead to this specific effect? First, the proportion of space occupied by these longitudinal cracks is far in excess of the elastic expansion of the metal. Second, it is during the recovery from the expansion caused jointly by pressure and heating that the cracks occur, and the doubled elastic transverse expansion would be followed by doubled transverse contraction, which would tend to make the longitudinal cracks smaller than the transverse. This principle, then, does not help explain the phenomenon.

45. *The Longitudinal Cracks on the Lands.*—On lands 1 and 3 of Fig. 16 there are cracks which, in their straightness and continuity, differ markedly from the transverse cracks, the irregularity of which argues that they are caused by contraction. These straight longitudinal cracks remind us of the scorings on the lands of the rear ring. Those on lands 1 and 3 and some of those on land 2 further resemble the rear-land cracks in being parallel to the axis of the gun instead of to the spiral rifling, but the most prominent of those on land 2 is parallel to the rifling. This suggests two different determinants of the direction of these cracks, the transverse contraction of the land for those parallel to the rifling, and scoring by the gases for those parallel to the axis.

46. *Relative Erosion of Land and Groove.*—The thickness of the metal in land and groove in my two rings is as follows:

| | Thickness at Land* | Thickness at Groove* |
|---|---|---|
| Forward ring | 0.4998 in. | 0.4666 in. |
| Rear ring | 0.4348 in. | 0.4311 in. |
| Difference | 0.0650 in. | 0.0355 in. |

* Each result here given is the average of four, each of which represents one of four grooves or lands taken at about 90° apart. Each of the original measurements was the maximum thickness of the land or groove examined. One can draw no close inference as to the relative erosion on land and groove from the photographs of Prof. Fay's rings (*Trans.* (1916), **56**, 470 and *U. S. Army Ordnance Department Report on Tests of Metals* (1913), Photo 2 after p. 134).

From this we infer that the erosion is about 85 per cent. more rapid on the top of the land than at the bottom of the groove, but yet there is evidently an important degree of erosion even at the bottom of the groove. These inferences are somewhat weakened by our not knowing that the initial dimensions of these two rings were identical.

47. *Relative Erosive Action of Driving Band and Gases.*—Erosion is generally referred chiefly to the gases and only in minor degree to the scraping action of the driving band. This is reasonable in view of the facts, first that the metal at any given transverse section is exposed incomparably longer to the rush of the gases than to the driving band, and second that the steel is incomparably hotter during most of its exposure to the gases than during its contact with the driving band.

The relation between the erosion of the lands and that of the grooves tallies with this belief. If the driving band did most of the erosion, then the lands should erode incomparably faster than the grooves, because the driving band, in becoming engraved upon the rifling, must press much harder on land than on groove. Consider a round rubber cork pressed into the neck of a bottle, ribbed on its inside.

We should not go so far as to suppose that the copper of the driving band does not reach the grooves, because the copper network in them seems to disprove this. That on the lower side of the bore might indeed have run into the cracks in a molten state during the contraction period, but that on the upper side cannot readily be explained thus. But the actual excess of erosion of land over groove, as shown in Sec. 46, is about that which we should expect from the higher temperature of the land, and the stronger rush of the gases past it.

Remember, too, that, after a few rounds, because of the progressive decrease of erosion from breech toward muzzle, the space into which the driving band has to fit is in effect tapered chokingly in this same direction, and further that because this erosion is greater on land than on groove, the taper or gradient too is stronger on land than on groove. This stronger gradient of the land would tend continuously to keep the copper out of contact with the grooves, or at least to lessen the pressure at that contact.

48. *Why the Copper Network Protrudes in the Grooves but not on the Lands.*—We have seen in Fig. 29 and in Sec. 38 the mechanism of the extrusion of a copper network from the cracks during the period of expansion, and its wiping away by the rush of gases. This wiping away is complete on the lands but incomplete in the grooves, implying that here the extrusion of copper continues after the rush of gases has ceased to be so violent as to wipe the copper away completely.

Before considering the reasons for this difference, let us recognize that though this extrusion begins immediately after the passage of the projectile, that is at moment $A$ of Fig. 2, so that the early extruded copper

is exposed to the most violent rush and highest temperature of the gases, it may well continue till this rush has spent itself. The inclination of the line $KL$ in that figure shows that the period of expansion, and consequent extrusion of copper, continues longer in the outer layers toward the roots of the cracks than at the very face of the bore, to $L$ instead of only to $K$. During all this period $KL$ the gases are cooling and shrinking fast, and thus fast losing their eroding power.

The first and chief of the reasons why the copper network remains extruded in the grooves, as is shown in groove 3 of Fig. 16, but does not in the lands, instead remaining flush with their surface, is that the period of expansion lasts later in the grooves than in the lands, because the groove-cracks are deeper than the land ones, and thus reach down into cooler metal, and because even at the bore face the grooves are less hot than the lands. For both these reasons acting cumulatively, the metal is cooler in groove than in land, and hence the period of expansion lasts longer in the grooves than in the lands.

A second reason is that, for given period, the velocity and temperature of the rushing gases are less on the grooves than on the lands.

To sum this up, the copper-extrusion lasts later and the erosion is feebler in groove than on land. On the lands the extrusion of the copper ceases when the erosive action is still strong enough to wipe the copper away flush with the steel, but in the grooves it lasts till this erosive action is no longer strong enough for this, and hence the last extruded copper persists as a dyke-network.

The contrast between the brightness of the lands and the oxide-black of the grooves explains itself in like manner by the greater erosive action on the face of the land than in the grooves.

I have assumed that the protruding copper network in the grooves which we find was formed toward the end of the last round fired, and that that formed in this period in any one round is effaced during the period $AE$, Fig. 2, of strong erosion in the next one. The reasons for this belief are so strong that we are reluctant to accept the alternative hypothesis, that either the protruding network or the black coating in the grooves is cumulative, and lasts from round to round.

49. *Why Some Cracks are not Filled Flush.*—The copper mouthful hypothesis leads us to expect every crack to be filled flush with copper at each round. We saw in Sec. 39 that the copper filling of any given crack was not a single firmly coherent mass. This hardly surprises us. When a crack opens during the period of cooling and contraction, its present copper contents may well become coated with oxide, carbon, or ash, of which the thinnest layer will suffice to prevent it from becoming firmly attached to the copper mouthful next bitten off, great though the pressure is which forces them together in the immediately following expansion period. Again, the copper may well be prevented in the same

way from becoming firmly attached to the steel walls of the crack. Moreover, the difference between the coefficients of contraction of steel and of copper might well tend to separate them during this contraction, and indeed the forcible opening of the jaws then detaches the copper from one face of its crack, and might weaken its hold on the other. Hence we are not surprised that certain cracks fail to retain their last mouthfuls of copper, which is all that is meant by their not being filled flush.

50. *Does the Cracking of the Face of the Bore Increase the Erosion?*—We naturally expect it to by increasing the heating up of the steel near the bore, by means of the heat of the gas which enters the cracks. I doubt whether it has an important effect in this respect, because the hardened layer does not follow down the sides of the cracks, as it would in case there were an important entry of hot gas into them (Sec. 34 and 39).

Indeed, the longitudinal and transverse cracks should, in a sense, lessen the tangential and longitudinal stresses tending to burst the gun. In a material which does not crack, tensile stresses, both longitudinal and tangential, would remain at the end of the cooling, of such intensity that, if unopposed, they would now create cracks the width of which would represent the contraction in that cooling. These stresses in an uncracked tube, and the cracks in a cracked one, in effect lessen the sum of the tangential and longitudinal stresses respectively tending to burst the gun, because they take up the expansion which the bore face undergoes because of its heating, and to that extent prevent it from adding to the sum of the stresses developed by the gas-pressure.

## III. Palliatives of Erosion

51. *What is the Temperature of the Metal which is Eroded?*—In the period, about 0.02 sec., between $A$ and $F$ of Fig. 2, hardening occurs to a depth of about 0.0047 in., or at the rate of about 0.23 in. per second. This means that the isotherm of effective hardening, say about 725°, migrates outward at about that rate.

In view of the slowness with which such an isotherm can be induced to travel into a mass of steel which we heat in a furnace at, say, 1000°, the temperature which the bore face itself must reach in order to induce so rapid an outward travel of this isotherm should be extremely high, because the rapidity of movement of an isotherm increases with the steepness of the thermal gradient behind it. It would be surprising if the temperature did not reach the solidus, about 1350° for gun-liner steel, and it may well reach the liquidus, or even a much higher temperature.

52. *Does Volatilization Cause an Appreciable Part of the Erosion?*—At these temperatures volatilization should be rapid, but in view of the

extremely brief time available one doubts whether it can account for a large proportion of the erosion. Of the assumed 0.02 sec. which the projectile remains in the gun after passing my forward ring, a certain fraction is consumed in heating the face of the bore to a temperature at which the rate of volatilization becomes considerable. But ignoring this, and admitting that a corresponding volatilization might go on for about the same length of time after the projectile leaves the gun, we have a total time of $0.02 \times 170 = 3.4$ sec. available for volatilization.

If we now recall our experience with the open-hearth process, in which molten steel lies boiling for hours well above its liquidus, and with the Bessemer process, in the old forms of which molten iron was exposed to currents of air for some 20 min. in a finely divided state and hence with a great extent of surface from which volatilization could occur, and if we further remember how small the loss referable directly to volatilization in these processes is, we must admit that the loss by volatilization in a total time of 3.4 sec. from the relatively small extent of surface offered by the interior of a gun liner could not form any considerable fraction of the quantity actually removed by erosion.

Moreover, the erosion in the powder chamber is reported to be negligible, "but a few thousandths of an inch," though here the time available for volatilization reaches its maximum. The fact that no erosion occurs in closed bombs points in the same direction, though less clearly, because the metal volatilized here might well re-condense later and thus leave no trace of having volatilized.

In brief, the evidence supports the natural expectation that the loss from volatilization should be inconsiderable.

53. *The Need of White-hardness.*—The driving band exerts very great pressure, but this is before the face of the bore has been heated highly by the exposure to the hot gases. The temperature of the bore face continues to rise rapidly after the driving band has passed, because of the now direct exposure to the hot gases.

Their velocity and pressure are so enormous that they probably scrape forward and detach not only the metal which is actually above the solidus, but also that which is at a temperature of great softness shortly below the solidus.

The pulling away of each particle implies overcoming the cohesion which holds it to those left behind, and hence the resistance of the metal to erosion should increase with its cohesion, and hence with its hardness, which is simply the name which we give to a varied group of manifestations of cohesion.

We set aside certain manifestations of the cohesion of the whole section of a test bar under such names as "tensile strength," "compressive," "shearing," "transverse," and "torsional strength," and we group together most or all of the remaining manifestations of cohesion as "hard-

ness" whether they are manifested in a scratching test such as that of Turner's sclerometer, or in a compressive ball test like Brinell's, or in a wear test, or in the rebound of a test hammer like Shore's, though these tests have little more in common with each other than they have with the determination of the tensile strength. It so happens that most of these manifestations of cohesion which we group together as "hardness" have to do chiefly with the superficial parts of the specimen, but this distinction is not applied consistently, as for instance when we say that an armor plate is "hard" from face to back. "Hardness," then, is a heterogeneous collection of unselected manifestations of cohesion.

Because resistance to wear is a manifestation of cohesion and thus related to the other forms of hardness and strength, we naturally infer that white hardness should oppose erosion, which after all is only a special form of wear.[40]

Because the rushing gases sweep forward not only the metal actually melted but also that which is erodibly soft, erosion is resisted not alone by a high melting point but also by what I venture to call "white-hardness," hardness at a white heat and even when near the solidus. White-hardness thus is a quality comparable with red-hardness.

54. *Other Properties Increasing the Resistance to Erosion.*—Because that which counts is the thickness of metal eroded at each round, and hence the thickness heated to erodible softness, erosion should be inversely not only as the melting point and the white-hardness, but also as:

1. The thermal conductivity.
2. The specific heat.
3. The density.

Great thermal conductivity increases the resistance to erosion because the heat which causes erodible softness is that which has not been conducted away to the outer and cooler parts of the gun liner, but instead has remained in the inner layers and raised their temperature.

That conductivity has time to play its rôle is proved by the fact that the hardened layer, which in my rear ring is about 0.005 in. (0.127 mm.), or about ten times as thick as the layer actually eroded in a given round, is heated by conduction to above $Ac_1$, say 725°. We have seen in Sec. 7 that this hardening recurs in each round, so that this appreciable thickness is heated in each round to above $Ac_1$. Of course, all of it, except the molecules actually forming the surface of the bore, receives its heat solely by conduction.

Great specific heat increases the resistance to erosion because it

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[40] In this connection see J. E. Hurst (Cast-Iron: with Special Reference to Engine Cylinders. *Engineering* (Jan. 19, 1917) **103,** 51) who finds that the strongest cast irons are also those which resist wear best in engine cylinders.

lowers the temperature to which the metal is raised by a given quantity of heat absorbed by it, that temperature being $\frac{H}{W \times Sp.\,Ht.}$, where $H$ = the quantity of heat available, $W$ = the mass of metal over which it is distributed, and $Sp.\ Ht.$ = its specific heat through the range of temperature in question.

Great density increases the resistance to erosion, because the greater the density the lower is the temperature to which a given thickness of metal is raised by a given quantity of heat.

55. *Evidence Supporting these Inferences.*—These inferences help to explain the apparent anomalies in the relative resistance to erosion of various metals. For instance, Table 3 arranges in the order of the melting point the results of Vieille's experiments on the erosion by volume of the various metals.[41]

Here the fact that copper erodes much less than its relatively low melting point would imply is readily explained by its great thermal conductivity, which explains also its excellent endurance of the trying conditions of the driving band, and its resistance to erosion when used for gun-vents.[42]

The erosion of silver exceeds that of copper by far more than is explained by the difference in their melting points, 1083° and 960°. They are substantially alike in conductivity. The excessive erosion of silver is readily explained by its low specific heat, only a little more than half that of copper.

That aluminum loses more than twice as much by volume as zinc, in spite of a decidedly higher melting point and much greater thermal conductivity, is explained in part by its low density, and in part by its softness. The product of its specific heat into its specific gravity, that is, its heat capacity per unit of volume, is materially less than that of zinc, 0.666 against 0.722.

Complete accord between the erosion and these physical properties is not to be expected, first because we cannot tell the ratio of the importance of the several properties, and second because the relation between any two metals with regard to any given one of these properties at the high temperature of erosion is likely to differ greatly from the relation between them at the room temperature, at which most of these properties have been determined.

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[41] Most of these experiments were made with powder containing 50 per cent. of nitroglycerine, and with cylindrical specimens 22 mm. (0.86 in.) in diameter, and 40 mm. (1.56 in.) long.

[42] See Abel and Maitland: *Journal of the Iron and Steel Institute* (No. 2, 1886), 467, who inform us that copper vents "resisted the wear of gas better than vents of either cast iron, wrought iron, or steel."

TABLE 3.—*Erosion of Various Metals, after Vieille, with Certain of Their Physical Properties*

| Metal | Loss by Erosion by Volume | Melting Point °C. | Brinell Hardness | Sp Gr | Specific Heat: Degrees C. From | to | | Thermal conductivity |
|---|---|---|---|---|---|---|---|---|
| Platinum | 59.1 | ...... | ..... | 21.4 | 0 | 300 | 0.03277 | 0 17 |
| Pure iron | 68 2 | 1530 | 75 | 7.86 | 0 | 500 | 0 1338 | 0.16 |
| Copper | 48 7 to 98.8 | 1083 | 74 | 8 9 | 15 | 338 | 0 09575 | 0.9 |
| Silver | 230.8 | 960 | 59 | 10 5 | 15 | 350 | 0.0576 | 1.0 |
| Aluminum | 2169 0<br>2307.0 | 659 | 38 | 2.7 | 20 | 508 | 0.2467 | 0.5 |
| Zinc | 1017 9 | 419 | 46 | 7.1 | 21 | 337 | 0.10173 | 0.26 |
| Tungsten | ........... | 3000 | 280 | 19 0 | 15 | 423 | 0.0375 | |
| Molybdenum | .......... | 2500 | 262 | ........... | 15 | 440 | 0.0740 | |
| Manganese steel | ........... | ...... | Initial 125<br>Final 540 | 7 83 for 13.75% Mn | 10 | 13 | 0.124 | 0.031 for 10% Mn |

NOTE TO TABLE.—The erosion appears to be that caused in a vent, made of the metal tested, by the passage of the powder gases from a bomb of 17.4-c.c. capacity, with a maximum pressure of about 2500 kg. per square centimeter (35,600 lb. per square inch).

The melting points are those given by the U. S. Bureau of Standards, *Circular* No. 35 (June 15, 1912), 2, except that of molybdenum, which is Fahrenwald's determination, *Trans.* (1916), **56**, 614.

The hardness is Brinell's (*Journal of the Iron and Steel Institute* (No. 1, 1901), **59**, 261) except that of tungsten and molybdenum which are Fahrenwald's (*op. cit.*, 617) and that of manganese steel of which the initial is my own (*Metallography of Steel and Cast Iron* (1916), 464) and the final that of Sir R. Hadfield and B. Hopkinson (*Trans.* 1914, **50**, 486, and *Journal of the Iron and Steel Institute* (No. 1, 1914), **89**, 112 and 124).

The density, specific heat, and thermal conductivity are from Landolt and Bornstein, Roth, 1912, except the specific heat of manganese steel, which is A. Crichton Mitchell's (*Transactions of the Royal Society of Edinburgh*, **35**, part IV, 952–4) and the specific gravity of manganese steel which is R. A. Hadfield's, *Proceedings, Institution Civil Engineers*, (1888), **93**, 61 and Howe, *The Metallurgy of Steel* (1890), 361.

This is strikingly true of hardness. For instance, to case-harden a pearlitic nickel steel may cause an extremely great increase in its hardness as measured in the cold and after air cooling, simply because the addition of the retarding influence of the carbon to that of the nickel causes the transformation to proceed in such a cooling only as far as the extremely hard martensitic state. But in a gun liner such a steel, when exposed to erosion, is heated up to the austenitic state, and the hardness of the martensitization which is acquired during the air cooling is thereby completely removed. Thus while the case-hardening increases the cold-hardness greatly it affects the white-hardness relatively slightly. This

tallies with the observation made by Taylor and White[43] and confirmed by Arnold[44] that cold-hardness is no measure of red-hardness.

What is true of hardness is probably true to a certain extent of the other properties. That copper, for instance, excels most other metals in conductivity in the cold does not prove that it does at whiteness.

56. *Evidence that the Melting Point is not the Sole Measure of Resistance to Erosion.*—We have seen evidence to this effect in the last section, in that the great gap of 447° between the melting point of iron and that of copper is accompanied by a relatively small increase in the erosion, and in that aluminum, with a melting point 240° above that of zinc, loses more than twice as much as it by erosion.

TABLE 4.—*Looseness of the Relation between Carbon Content and Erodibility of Gun Steel. Royal Gun Factory Tests*

| Number | Mean Order Ten Observations | Percentage Carbon | Percentage Manganese | Percentage Silicon |
|---|---|---|---|---|
| 1 | 1.9 | 0 219 | 0.529 | 0.107 |
| 2 | 2.1 | 0.233 | not determined | not determined |
| 3 | 2.5 | 0.414 | 0.086 | 0 059 |
| 4 | 4.1 | 0.333 | 0.655 | 0.093 |
| 5 | 6.0 | 0 216 | 0.475 | 0.054 |
| 6 | 6.2 | 0.160 | 0.320 | 0.042 |
| 7 | 6.4 | 0.172 | 0.255 | 0.119 |
| 8 | 7.9 | 0.391 | 0.780 | 0.064 |
| 9 | 8.0 | 0.520 | 1.050 | not determined |
| 10 | 9.8 | 0.241 | 0.664 | 0.068 |
| 11 | 11.0 | 0.347 | 0.407 | 0.051 |
| 12 | 12.0 | 0.182 | 0.050 | 0.330 |
| 13 | 13.0 | 1.144 | not determined | not determined |

NOTE TO TABLE.—Thirteen 2¼-in. rifled steel barrels were tested by firing from each 100 rounds of 10¼ lb. of pebble powder and 6 lb. shot. These barrels were screwed into the mouth of a 22-cwt. breech loader. After every 25 rounds each of five skilled observers took ten impressions of the interior of each barrel with the average results given in Column 2. The erosion in No. 11 is about double that in No. 1. *Journal of the Iron and Steel Institute* (No. 2, 1886), 469.

Further evidence to this same effect is given by three series of tests, those of the Royal Gun Factory,[45] in which 2¼-in. (57.15-mm.) gun barrels

[43] F. W. Taylor: The Art of Cutting Metals, *Supplement to Proceedings of American Society of Mechanical Engineers* (November, 1906), 192–3.

[44] J. O. Arnold: Note on the Relation Between the Cutting Efficiencies of Tool Steel and Their Brinell or Scleroscope Hardnesses. *Journal of the Iron and Steel Institute* (No. 1, 1916), **93**, 102. "The Brinell hardness of a properly hardened tool is an almost negligible factor of efficiency."

[45] See Abel and Maitland: *Journal of the Iron and Steel Institute* (No. 2, 1886), 468.

were tested; those of Vieille, in which cylinders 0.87 in. in diameter and 1.57 in. long (22 by 40 mm.) were used; and those of the U. S. Naval Proving Ground, which determined the loss by erosion of plugs of various materials, each forming the vent of a bomb in which the explosion took place. Of these the Royal Gun Factory tests carry by far the greatest weight, because they were repetitive, firing 100 rounds in each barrel with 6-lb. shot, thus reproducing the conditions of service, in which the erosion in each round is of the metal hardened and cracked in the previous rounds. In the other tests the erosion is caused by a single explosion without any projectile, that is to say by the gases only.

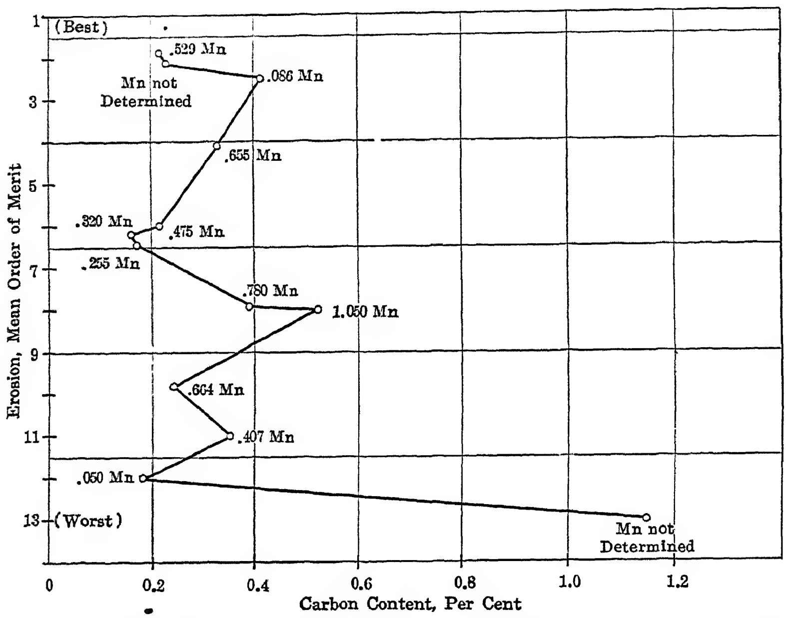


FIG. 31.—IRRELATION BETWEEN CARBON CONTENT AND EROSION IN TESTS OF ROYAL GUN FACTORY.

In the Royal Gun Factory tests the third best of the thirteen steels tested contained no less than 0.414 per cent. of carbon, and therefore had a solidus of about 1350° C. Of the ten steels which it excelled in its resistance to erosion, no less than eight had less carbon, running from 0.16 and 0.172 to 0.391, and hence with a solidus temperature from 76° and 68° to 10° higher. These materially higher melting points thus are accompanied by an increase in erosion.

The irrelation between the melting point and the erosion is shown graphically in Fig. 31, which represents these same tests.

In Vieille's tests nickel, with a melting point of 1452°, eroded very much more than gun steel of about 0.15 per cent. of carbon and therefore with a melting point (solidus) about the same as that of nickel.

In the Naval Proving Ground tests "iron," evidently very pure wrought iron, was fifth in order of merit, and thus worse than much more fusible steels of which one with 0.276 per cent. of carbon, had a solidus about 130° lower.

According to our present evidence the least erosion occurs with a plain carbon steel of medium-carbon content, somewhere about 0.35 per cent. A reasonable explanation is that this gives a good combination of a high melting point with white-hardness. As the carbon content increases the melting point falls but the white-hardness increases, as is shown by the greater power needed for rolling and hammering high-carbon than low-carbon steels.[46] We may infer that when the carbon content increases beyond about 0.35 per cent., the harm done by further lowering of the melting point outweighs the good done by the increased white-hardness, and on the other hand that when it decreases below 0.35 per cent. the influence of the loss of white-hardness outweighs that of the rise in the melting point.

57. *Possible Palliatives of Erosion.*—It seems too much to hope that the velocity and pressure of the gas-stream in the bore shall be great enough to give the projectile its enormous velocity in a trifling fraction of a second, and yet shall not be able to tear away some solid particles from the face of the bore. Add to these the very high temperature developed, and our wonder rather is that the erosion is not greater.

Conceivable ways of lessening erosion are:

1. To reduce the temperature by devising an explosive which shall deliver more of its energy in the form of pressure and less in the form of temperature. On this point the metallurgist may well hold his peace, especially because a large part of the present pressure must be a result of the high temperature.

2. By lubrication, for instance with graphite, which should be affected but little even by this high temperature.

3. By devising ready means of drawing and replacing the gun liner. To the layman this seems incomparably the most promising means. With electricity as a ready means of heating and expanding the body of the gun, and carbonic snow and liquid air as ready means of contracting the liner, and with graphite to lubricate it, surely our present mechanical engineers should be able to pull an old liner out and push a new one in, in view of the antiquity of the invention of shrinking a tire upon a wheel.

4. By using a less erodible metal. This is the only palliative which I should venture to discuss.

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[46] See O. Fuchs: Der Einfluss von Temperatur und mechanischer Arbeit beim Pressschmeiden von Flusseisen und Stahl. *Zeitschrift des Vereines Deutscher Ingenieure,* (1915), 915–918, and Influence de la température et du travail mécanique lors du laminage de la fonte et de l'acier. *Revue de Métallurgie, Extraits* (1916), **13,** 200.

58. *Possible Less Erodible Alloys.*—Because a high melting point is evidently of very great importance, we naturally turn to alloys of iron with metals more infusible than it. Of these there are only four, tungsten, molybdenum, vanadium, and titanium, which are to be had in quantity and at a price that makes them worth considering. I am informed that, if these were sought in quantities sufficient to form say 10 per cent. of an alloy from which thousands of heavy gun liners were to be made, their prices would be very roughly as follows:[47]

| | | Present Price per Pound | Normal Price per Pound |
|---|---|---|---|
| Tungsten | melting point about 3,000° C. | \$2.50 to 2.75 | about \$1.00 |
| Molybdenum | melting point about 2,500 | 4.00 to 5.75 | \$1.30 to 2.00 |
| Titanium | melting point about 1,800 | 3.00 | |
| Vanadium | melting point about 1,750 | 2.63 | \$2.25 |

It is reported most persistently and as persistently denied that molybdenum steel is used by our Allies for gun tubes.

A like report concerning enemy ordnance led to my having Booth, Garrett & Blair analyze pieces cut from two German guns, with the following results:

10.2-cm. (4-in.) German gun received through Mr. Hudson Maxim.

| | |
|---|---|
| Carbon | 0.372 |
| Silicon | .199 |
| Manganese | .368 |
| Phosphorus | .017 |
| Sulphur | .025 |
| Nickel | 2.870 |
| Chromium | 1.270 |

15-cm. (5.9-in.) German howitzer received through Col. N. T. Belaiew, Russian Artillery.

| | |
|---|---|
| Carbon | 0.430 |
| Silicon | .229 |
| Manganese | .218 |
| Phosphorus | .016 |
| Sulphur | .045 |
| Nickel | 2.670 |
| Copper | .136 |
| Chromium | 1.350 |
| Cobalt | .040 |

The former of these represents a captured 10.2-cm. (4-in.) German gun, of which I have received five rings from the French High Commission through the kindness of Mr. Hudson Maxim and of Brigadier General William Crozier, late Chief of Ordnance, U. S. Army.

The second analysis represents a small fragment of a captured 15-cm. (5.9-in) German howitzer received from Colonel N. T. Belaiew of the

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[47] For these prices I am indebted to E. J. Lavino & Co., Mr. G. L. Norris, of the American Vanadium Co., the Primos Chemical Co., the Titanium Alloy Manufacturing Co., and the Vanadium Alloys Steel Co. These metals are more abundant than might at first be supposed, as is shown by the ability of the Primos Chemical Co. alone to supply annually about 2,000,000 lb. of tungsten, 1,000,000 lb. of vanadium and 1,000,000 lb. of molybdenum. The price of \$2.63 per pound for vanadium is a special one for the United States Government. The market price is given as about \$3 per pound.

Russian Artillery, by the permission of Mr. Byron, Director of British Artillery.

The absence of the relatively infusible elements, tungsten, vanadium, and molybdenum, is interesting in the present connection. The purity of the steels, especially that of the first, is in strong contrast to the remarkable impurity of German projectiles analyzed by Stead,[48] and suggests that this is deliberate rather than unavoidable. It tends to strengthen the layman's natural inferences that though a gun should be of the very strongest material, the projectile need not be, and it reminds us of the work done with chilled cast-iron projectiles, and even with lead.

Unfortunately our present evidence indicates that additions of these relatively infusible metals, with the possible exception of tungsten, increase the fusibility of iron.[49] Moreover alloys of iron with large quantities of these metals are very costly, and in most cases very difficult to machine and very prone to oxidize. Here, then, the outlook is far from encouraging.

59. *Tungsten.*—One of the most intelligent makers of high-speed tool steel informs me that tungsten raises the "pouring temperature" at the rate of about 11° C. per 1 per cent. of tungsten.[50] This does not necessarily conflict with the inference which Guertler draws that "The melting point of iron should be lowered little at the first by tungsten."[51]

60. *Molybdenum.*—Guertler's "preliminary and hypothetical" diagram of the iron molybdenum alloys[52] implies that even the liquidus is lowered by an addition of some 22 per cent. of molybdenum to about 80° C. below the melting point of pure iron, and that it is not till the molybdenum content reaches about 42 per cent. that the liquidus again rises as high as the melting point of iron.

61. *Titanium* is even less encouraging, for additions of it up to 13 per cent. lower the liquidus rapidly and the solidus extremely rapidly, bringing it down to 1300°, or 230° C. (446° F.) below that of pure iron, when the titanium content reaches 6 per cent. The solidus remains at this temperature at least till the titanium content reaches 19.90 per cent.[53]

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[48] *Proceedings*, Cleveland Institution of Engineers, Session 1, 1915-16, p. 51. Nine out of the 22 high-explosive German shells contained from 0.06 to 0.105 per cent. of phosphorus.

[49] This tallies with Admiral Earle's assertion that steels with as much as 43 per cent. of nickel and 14 per cent. of tungsten "are the worst that have been tested" as regards erosion. *Trans.* (1916), **56,** 494.

[50] "The pouring temperature of high-speed steel carrying about 15 per cent. of tungsten is approximately 300° F. above the pouring temperature of carbon tool-steel of the same carbon content."

[51] *Metallographie* (1912), **1,** 383.

[52] *Idem*, 372.

[53] J. Lamort: Über Titaneisenlegierungen. *Ferrum* (1914), **11,** 225–234. *Revue de Métallurgie, Extraits* (1916), **13,** 81.

62. *Vanadium*, too, lowers the melting point of iron, though by no means so rapidly. A vanadium content of about 33 per cent. is reported to lower both liquidus and solidus to about 1440°.[54]

63. *Manganese steel* has shown surprising endurance of conditions closely like those of the lands, in its use as a pipe ball in lap-welding tubes. Here the white-hot steel skelp under very great pressure is scraped very rapidly over the face of the ball, much as the white-hot gases of the gun rush past the surface of the land. In both cases we have rapid frictional motion under great pressure at a very high temperature.

Why manganese steel is so endurant under these specific conditions we do not know. Indeed, its relatively low thermal conductivity and specific heat might at first seem to work against its endurance, if we should forget that these properties as we determine them in other steels are those of the pearlitic state, and hence give little suggestion of what they are in the austenitic state in which erosion is resisted.

Against manganese steel must be weighed its proneness to oxidize, and the very great cost of rifling it.

64. *Possible Application of Unsoftenable Steels.*—If any of these steels which cannot be softened in order to facilitate machining, tungsten, molybdenum, and manganese steels, should show great resistance to erosion in preliminary tests, a possible way of using them would be to restrict them to the rear part of the gun, where the erosion is most rapid, to cast the rifling in, and to take a very light finishing cut with an emery wheel. If the life of great guns is as brief as we have sometimes been told, then if we had a substance far more resistant to erosion than our present gun steel, but very expensive to machine, it might be wiser to use it in the rear of the gun, and machine it only roughly to dimensions, in view of the fact that in present practice the rifling retains its initial accuracy for only relatively few rounds, after which it degenerates greatly through erosion. The fact that the gun gives by far the greater part of its useful service after it has lost its initial accuracy of rifling proves that this accuracy is not essential to usefulness.

## IV. Summary

The paper is divided into three parts, which treat severally of the hardening of the bore by its rapid cooling by the outer metal, the cumulative cracking of the bore, and possible palliatives of erosion.

The explanations of the various phenomena are based chiefly on a hypothetical "temperature cycle" (Sec. 8), or course of heating and expansion in each round.

[54] Vogel and Tammann, quoted by Guertler: *Metallographie* (1912), **1**, 387.

At any section the cycle begins with the passage of the driving band of the projectile, from which the lips of each crack, hardened in the preceding round and still cool, bite off a mouthful of copper (Sec. 38), thus sealing themselves. Next comes extremely rapid heating, expansion, and closing of the cracks, with extrusion of much of the copper. A thin layer of the bore face is melted and swept forward by the rush of the gases, together with so much of the metal outside it as is heated to erodible mobility, the metal removed from any one section being thus replaced in part by that swept forward from rear sections, and this forward sweeping layer keeping all cracks sealed till the bore-face metal again hardens (Sec. 41). With this forward sweeping probably goes the entrainment of some of the molten metal in the form of a mist, its bodily removal from the gun (Sec. 52), and choke-boring. Thus the erosion at any given section consists first of this misty entrainment, and second of the excess of the metal swept forward from that section over that swept to it.

In the heating period, brief as it is, some of the initially distinct particles of ferrite and cementite in a very thin layer adjoining the bore face merge to form austenite. The following cooling by conduction into the cold outer metal is so rapid that this austenite is caught in the state of martensite and is thus hardened (Sec. 6 and 9). It is next annealed during the heating period of the next round and in part eroded and re-hardened in the following cooling. Thus the hardening is repetitive (Sec. 7).

The thickness of this hardened layer increases asymptotically from round to round, because the heating in each round extends beyond the layer actually hardened in the ensuing cooling, and in thus extending begins the merging of the ferrite and cementite beyond the layer hardened, and thus facilitates further merging and thus hardening in the next round (Sec. 11).

Though we might well expect the powder gases to carburize the bore face and thus to increase the hardening, on closer examination we see that appreciable carburization is very improbable because of the brevity of the total time. It is proved to be lacking in my forward ring (Sec. 24).

Hardening may hasten erosion, by causing particles to flake off from the bore face while it is heating up (Sec. 25).

In the cooling of each round the layer which has been fluid or at least pasty cools and hence contracts much more than the outer and cooler metal with which it is integrally united, and thus becomes in effect stretched by the resistance of this strong outer cold metal beyond its ability to retain continuity, and thus cracks (Sec. 26).

In the grooves the expansion has to be axisward solely, and the cracking in the subsequent cooling is, therefore, about the same longitudinally as circumferentially (Sec. 33). But the forward rush of the gases

exaggerates the longitudinal cracks, so that the longitudinal ones are more prominent than the transverse. The expansion of the lands occurs both axisward and circumferentially, and the corresponding circumferential contraction in large part relieves the tendency to form longitudinal cracks, with the result that the main cracks are transverse (Sec. 35).

The copper stopping of each crack, and the oxidation of its sides during the period of contraction, cause the cracking to be cumulative, so that the total width of the cracks, including their copper stoppings, is about 10 times that corresponding to the contraction occurring in a single round (Sec. 37 and 38). The transverse cracks on the strongly pressed lands are so much more prominent than those in the grooves, and have such an approach to even spacing, as to suggest strongly that they are exaggerated by chattering (Sec. 43).

In the forward lands many of the cracks are filled flush with copper. The rest are black and not filled flush. A copper network projects from the forward grooves. The difference is referred jointly to the continuation of the expansion in the grooves till a later period than in the lands, and to less violence of the gas-rush in the grooves than on the lands, with the result that all the copper ejected by the expansion from the land cracks is wiped away by the gases, while the latest extruded from the groove cracks persists (Sec. 48).

The cracking of the bore probably neither hastens erosion nor weakens the gun (Sec. 50).

The temperature of the metal eroded probably rises at least to the solidus, if not to the liquidus or even beyond it, but some of the metal eroded is probably between the solidus and a lower temperature, that of erodible mobility (Sec. 51).

Volatilization probably does not contribute greatly to erosion (Sec. 52).

White-hardness (Sec. 53) and also great thermal conductivity, specific heat, and density, should lessen erosion (Sec. 54).

Of the palliatives of erosion, the replacement of the liner seems the most promising (Sec. 57).

Though experiments with other alloys of high melting point should be tried, the prospect of success is not great, because the available metals more infusible than iron are thought to yield alloys less infusible than it, and also difficult to machine, costly, and prone to oxidize (Sec. 58).

The success of manganese steel in pipe balls in resisting conditions closely like those of gun liners suggests experimenting with it (Sec. 63).

## Discussion

Hudson Maxim, Brooklyn, N. Y.—Immediately after the creation of the Naval Consulting Board, Admiral Strauss, Chief of the Bureau of Ordnance of the Navy, told members of the Board that the most im-

portant problem for the scientists to solve was erosion in large guns. The Navy, for years, had been working on the problem with marked success, but there was much yet remaining to be accomplished.

As Chairman of the Committee on Ordnance and Explosives of the Naval Consulting Board, I communicated with several experts in the manufacture of steel, principal among whom was Prof. Howe. He generously agreed to help in every way possible, and the work that he has done and the time that he has spent upon it can be appreciated only by reading his paper.

Prof. Howe asked for some rings from a certain 14-in. gun which Admiral Strauss had mentioned in particular, and in which, he had informed us, at each shot the bore of the gun was enlarged about 0.001 in., and that after a relatively small number of rounds it was considerably eroded. It has puzzled scientists for a long time to account for the peculiar erosive effect produced by the powder gases in a gun, and what Prof. Howe has done in his paper is worthy of Sherlock Holmes. He is a veritable Sherlock Holmes of science.

The cracking of the surface of the bore has always been a great puzzle. The cracks are not produced directly by the erosion; he has discovered that they cannot be, because the bottoms of the cracks are not rounded. One of the most brilliant discoveries that he has made is that as the cracks in the gun open after each successive shot they take a mouthful of copper out of the driving ring at the next shot and hold it; that on the lands of the gun, the wash of the gases takes this copper off smooth with the surface of the lands; while in the grooves of the gun, the wash of the gases being less, the copper is seen in a sort of network extruded from the cracks after the gun cools.

He has also discovered that when a shot is fired, the wash of the metal covers over and chokes up the mouth of the cracks, and then when the projectile has left the gun and it cools, the cracking again occurs. He has shown that the cracking occurs mainly from the shrinking of the heated, hardened surface away from the outer and cooler body of the gun. He gave a very pretty simile of exactly how that peculiar effect occurs. When the gun is fired, the hot gases carry the fused surface of the metal forward just as the wind on the sea carries the water forward on the surface; and you might also perhaps get a similar idea of the peculiar action by noticing how the wind blows water that has melted on the surface of ice; he has also shown that the copper and the steel are actually washed forward and to a considerable extent deposited in the forward part of the gun, choking it somewhat. He has also discovered that there are very good grounds for the belief that there is a certain amount of carburizing from carbonic oxide.

In regard to the palliatives that he suggests, Prof. Howe thinks that white-hardness would be a good remedy, that a white-hard steel, that is

to say, a steel which would be hard at a white heat, would make a surface which would minimize erosion. He also refers to the facility with which guns can be re-lined.

As to the latter remedy for erosion; it costs $17,287 to re-line a 14-in. gun, and takes about 3 weeks to dismount it, re-line it, and put it back, but they have a lot of spare guns always on hand, so that when a ship comes in and changes its guns, it does not necessarily have to wait for its own guns to be re-lined. Since the knowledge was obtained on which Prof. Howe's paper was based, the Navy Department has made some very important investigations and has made very rapid progress in overcoming the problem of erosion. To what extent they have succeeded, I am not at liberty to describe, but I am permitted to say that they have already fired over 250 rounds from a 14-in. gun and it still fires accurately. A ship's guns can therefore fire twice as many rounds without serious erosion as the ship can carry.

When this matter was taken up in the early stages of this investigation, we had an idea that the Germans had gone away ahead of us in the production of non-erosive steel for their guns, but we have found that if the Germans win this war, it will not be because they have better gun steel than we have, but it will be in spite of the fact that we have far better steel than they, and better guns, and I hope it will prove, better men behind them.

One thing more I wish to mention is one of the most important features of the problem: the powder charge in a gun. Last year I referred to the importance of the powder and mentioned the improvement made by the duPont company in rifle powder. Also, that prior to the discovery of that peculiar rifle powder, a shoulder gun could be fired only from 3000 to 5000 rounds with the best powder before it was so badly eroded that the balls would tumble. The duPonts have developed a powder with which a shoulder gun has been fired more than 20,000 rounds without losing its accuracy. The smokeless powder that has been developed by the Navy is made of pure nitrocellulose, a modification of the nitrocellulose discovered by Mendeléeff. This nitrocellulose makes a very excellent smokeless powder and one much less erosive than any of the powders used abroad in large guns.

A great many persons have an idea that when a gun is fired, the powder actually explodes in the sense that a high explosive explodes. It does not do anything of the sort. It merely burns like a piece of wood in the grate, only somewhat faster. The time that a projectile remains in a big gun after the powder charge is ignited is about $\frac{1}{50}$ or $\frac{1}{60}$ sec. In that time the whole 380 lb. of smokeless powder in a 14-in. gun has to be consumed. It is necessary to have hot gases in order to give the projectile the required velocity. Suggestions have been made, and numerous experiments have been tried, to cool the gases in order to lessen the

erosive effect on the bore of the gun. That will not do beyond a certain extent, and there is so much carbon monoxide produced and so little carbon dioxide in the combustion of our pure nitrocellulose smokeless powder that we have nearly reached the minimum of the coolness desirable in the gases. In one of the long Navy 12-in. guns, the heat sacrificed by the gases in giving the projectile its muzzle energy of, say, 50,000 ft.-tons, is equal to that required to melt something like 750 lb. of cast iron, so that the work in propelling the projectile from the gun is considerable, and it is the chief cooler of the gases.

If the gases were perfectly cool, there would still be erosion, even if it were possible to use gases at the temperature of the atmosphere, for the reason that there is such terrific vortex action, the gases traveling at such high velocity and having such great density. A meteorite, when it strikes the outer atmosphere, gets hot enough to ignite and burn up in the oxygen of the air as it passes through it. Now the speed of a meteorite is not much greater than the speed of the gases in the vortex motion that occurs in the bore of a gun.

The duPont company acquired an invention of mine—excuse me for referring to an invention of my own—for the multiperforation of smokeless powder, designed to burn progressively in the bore of a gun. There are two ways in which powder can be made to burn in a gun. The pure nitrocellulose colloid of which our smokeless powder is composed burns with such comparative slowness in a gun that, in order to employ a charge of sufficient mass to get the required velocity without prohibitive pressures, it is necessary that the powder be multiperforated so as to increase the initial surface presented to the flame of ignition, or nitroglycerin or some other oxidizing agent must be added to make it burn more rapidly. The simplest method is to multiperforate it.

I do not hold a brief for the duPonts, and do not own a dollar's worth of stock in the company, but they are patriotic people, and have always given the Government the full benefit, without charge, of every process that they have ever developed at their place, or have ever acquired. No matter whether it is something they have bought or something they themselves have developed, the Government has had the free use of everything, all their apparatus, their machinery, their advice, their efforts, and everything that it wants, free and without price.

Ralph Earle* and N. W. Pickering† (written discussion):

1. Prof. Howe's paper treats of the erosion of guns principally from the standpoint of the erodible mobility of metal, showing the temperatures to which the surface of the metal is subjected, the resultant

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† Lieutenant Commander, U. S. N., in charge of heavy gun design, Bureau of Ordnance.

liquefaction of the surface, and the scouring away of this metal by the rush of gases. This same fact is set forth in the Ordnance Bureau's *Pamphlet* No. 412, issued in October, 1912, although the subject is not gone into at such great length. Prof. Howe further shows that heat cracks are the result of this same liquefaction and sudden cooling. This point has also been stated in *Pamphlet* No. 412. The original feature in Prof. Howe's paper is the coppering caused by the cracks biting off pieces of the copper band. This appears logical, but unimportant. The copper that enters these cracks certainly has no choking effect in the bore of a gun, and therefore can be entirely omitted as affecting gun ballistics. The theory advanced is most interesting and probably true.

2. On page 515, it is stated that the usual form of rifling is of the "hook section." This is an error, as all recent guns have the ribbed rifling. Prof. Howe's theory that heat cracks are deepened by the accumulation of copper cannot be reconciled to the fact that in many guns the heat cracks are most apparent in the powder chamber. A recent case of this is the *Wisconsin*, in which deep heat cracks are noted for some distance abaft the compression slope. These cracks are certainly not subjected to the friction of the rotating band, and therefore cannot be caused by the insertion of copper in the crack. It is also noted that large heat cracks develop in large drop-forging dies, and that in order to prevent an increase in the size of cracks copper is peened in, giving a smooth surface and preventing spreading of the crack.

3. It is not believed that the transverse cracks can rightly be called chatter marks. A shell fits exceedingly tightly in the bore, and is moved with such an irresistible force that it certainly cannot be considered as chattering as it moves down the bore of the gun. Furthermore, the fact that transverse cracks are just as apparent in the powder chamber as in the rifled part of a gun would prove that chattering could not be the cause of transverse cracks.

4. Prof. Howe does not explain the reason why the erosion or bore enlargement per round is not uniform. He apparently assumes that the same amount of metal is carried away at each discharge of the gun. However, all curves show this not to be the case, and it is known that the lands in the breech end of a new gun are worn down rapidly for the first few rounds and much less, per round, toward the end of the gun's life. It is generally considered that erosion is principally due to the following reasons: (*a*) The gradual enlargement of the bore and smooth wearing away of the surface by the action of the gas in rear of the projectile. This varies as described above. (*b*) The irregular wearing away of the surface of the bore in holes and gutters, which is called scoring, and is mainly due to imperfect windage and sealing of the bore. Prof. Howe does not treat of this second condition.

5. The paper suggests that erosion be corrected by placing a new

liner in a gun. This method has been followed by the Bureau for approximately 10 years with perfect success, and the Bureau agrees with Prof. Howe that this is the best means of combating erosion. However, recent experiments have shown that careful oiling of the bore of a gun lengthens its life. The life of a small gun was increased 85 per cent. in this manner. It is strange that the use of graphite as a powder blown into a gun has not proven to have any particular advantage, while sperm oil, carefully rubbed in, is very efficient.

6. We do not agree with Prof. Howe's conclusion that the cracking of the bore neither hastens erosion nor weakens the gun. Any crack that tends to increase its depth with each round may considerably weaken a gun that is built without a liner. This crack may continue a considerable distance into the tube with the result of weakening it. The tube is one of the main strength members of the gun. If it is admitted that the opening of the heat crack is completely closed with fluid metal by the rush of gases, then it must be admitted that heat cracks are not a cause of erosion; otherwise they are a very serious cause of erosion, because they offer easy points of attack for the gases, and scoring or guttering will occur in their immediate vicinity. It has been found that small sand splits in a powder chamber, or in the origin of the rifling, are quickly attacked by the erosive action of gases, with consequent dishing or scoring of the bore. It seems quite probable that this same effect would take place on heat cracks unless it can be conclusively proved that Prof. Howe's theory, that the mouth of these cracks is welded during firing, is correct.

7. The theory of specific heat and heat transmission is probably applicable to erosion. However, it is impossible to put this entirely into practice. Metals for use in guns must be considered in regard to their elastic limit, elongation, etc., and this necessarily prevents the use of certain metals, the thermal conductivity and specific heat of which would otherwise recommend them for guns on account of their erosive resistance.

Henry D. Hibbard, Plainfield, N. J. (written discussion*).—The two suggested means for prolonging the life of a gun are: first, to reduce the temperature of the powder gases, and, second, to manufacture a steel having a higher fusion point and better resistance to the erosion. The possibility of attaining the last-mentioned feature is by no means hopeless when we remember that we have at hand such metals as tungsten, molybdenum, tantalum, vanadium, and titanium.

For steel to be eroded, it is not necessary for it to be raised to the fusion temperature. If heated to, say, 1200° or 1300° C., the steel may conceivably be soft enough to be swept away by the heavy, swiftly rushing gases, heavy because they are under great pressure.

* Received Feb. 15, 1918.

It would be extremely desirable and helpful if a public record could be compiled of all the various efforts which have been attempted for overcoming erosion since high-powered guns came into general use 30 years ago. Negative results would be fully as useful as positive results. It would be particularly valuable if the makers of the steel used in fabricating each gun would give a detailed history of the production of that particular steel. Without that knowledge, the result of experiments have little meaning, and much of the work of the experimenters will be wasted.

J. S. Unger,* Pittsburgh, Pa. (written discussion).—Prof. Howe's paper deals with the strictly technical side of this question. I will try to present the subject from its practical side.

A 14-in. gun costs about $60,000 and has to be relined after, say, 170 shots have been fired, its actual firing life thus having been 10 sec. The cost of relining the gun once is about $18,000. During war, or when the gun is in actual service, it must be in the shop for repairs approximately 40 per cent. of the time. If the 14-in. gun fired 170 shots during the first 7 months of a year, it would be in the shops for the following 5 months. When this is considered, the importance of the practical side of the problem must be evident.

In order to have the tube of an unlined gun, or the liner of a lined gun, which are the parts affected by erosion, accepted by the purchaser, it must conform to these requirements:

1. The ingot must be made by approved methods.

2. The ingot must be of such size as to permit of a reduction in cross-section, during forging, of 4 to 1 when compared with the finished forging.

3. The forging must be annealed, then rough bored and turned, then heated and quenched in a cooling medium, followed by a final heating or tempering treatment at a lower temperature than the preceding treatments.

4. At each end of the treated forging four tangential specimens for tensile tests, 90° apart, are taken, which must show the following qualities, for a 14-in. gun:

| | |
|---|---|
| Elastic limit | 46,000 lb. per square inch. |
| Ultimate strength | 86,000 lb. per square inch. |
| Elongation in 2 in. | 18 per cent. |
| Reduction of area | 30 per cent. |

5. The steel must be readily machined.

6. The finished gun must stand a proof or firing test, or be subjected to an internal gas pressure of about 35,000 lb. per square inch, without deformation.

Other requirements could be given, but these are the principal ones.

* Manager, Central Research Bureau, Carnegie Steel Co.

They indicate that a plain carbon steel of about 0.50 per cent. carbon, or an alloy steel of about 0.40 per cent. carbon are the only materials available at this time which will meet the preceding requirements.

Erosion has been attributed to many causes, the principal ones being: high temperature, high pressure, mechanical abrasion by the projectile in the bore of the gun, rush of gas ahead of projectile, gas at high velocity behind the projectile, and the sand-blast effect of the grains of powder.

Without attempting to specify which cause is responsible, they may produce carburization of the metal, hardening and cracking of the surface, wearing or washing away of the metal, depositing of copper, iron, or their oxides in the cracks, and the production of a different macroscopic and microscopic structure in the part affected. Many of these results have been known for a number of years, but this knowledge has been of little direct value in correcting the trouble.

As pointed out by Prof. Howe, we are interested in the solution of the problem from one side only, the use of a metal or alloy which is less erodible than steel.

An examination of the results of actual experiments made with different metals shows results which at times are conflicting. We find that a Parson manganese bronze and a cast iron have given poor results when compared with steel under actual firing tests. Aluminum bronze in a small arm offered good resistance to erosion, but failed on account of lack of strength. Alloys used in tests in bombs were poorer than steel. Three steels of 0.30, 0.40 and 0.90 per cent. carbon, when used as liners in a gun with modern powder chamber pressures, showed that the higher the carbon or the hardness, the greater the erosion. Three steels of about 0.22 per cent. carbon, and as near the same chemical composition in other respects as it is possible to produce commercially, ranked first, fifth and tenth in their resistance to erosion in a firing test. Wrought iron and soft steels appear to offer good resistance to erosion in some tests, but such material will not meet the requirements of the present specifications.

Copper vents fastened in the body of a gun, which permitted the escape of gases only, gave good results, but not comparable with the effects produced in the bore of gun which resisted the gases and the mechanical abrasion. Plugs of various metals fastened in the orifice of bombs and subjected to heat and gas pressure were not satisfactory. Plugs inserted in the breech blocks of guns were in the same class. The erosion in the powder chamber of a gun is very small when compared with that at the origin of the rifling. The effect of forging the steel seemed to give some promise, as the greater the work done in forging, the better the resistance, until it was shown that when a great deal of work was done, the steel became inferior in its resistance to erosion.

Any substitute for a regular steel liner must not crush or burst during

service. Bursting may be prevented by supporting the lining material by the tube and jacket. It must have sufficient longitudinal strength to prevent rupture, and consequent blowing out of the gun. This might be prevented by proper support of the surrounding parts. *It must be readily machined.* If a substitute metal or alloy be found, which gives better resistance to erosion than steel, its cost is determined by the extra life it has beyond that of steel. This would be largely modified, however, by the advantage of possessing, in a case of great emergency, a gun with a much longer firing life.

Tungsten and molybdenum have been suggested as substitutes, but they are not easily machined, particularly tungsten, even when in the ductile condition. To prepare ductile tungsten in masses sufficiently large to be used for even a part of a liner would be almost impossible. Both metals oxidize easily at a red heat, and each absorbs carbon under proper conditions, becoming brittle. Their alloys with aluminum or copper, or with each other, may show better results than the metals alone.

Alloys of nickel and chromium are commercially produced, and could be obtained in sizes from which liners could be made. These alloys have properties, when subjected to high temperatures, which recommend them. A roll for a rolling mill is used under conditions of high temperature, pressure, and abrasion, resembling the conditions in firing a gun. When a comparison is made of a steel plate-mill roll with a cast-iron roll, or a chilled cast-iron roll, with or without the addition of nickel and chromium, it is found that the chilled-iron roll has a much longer life, before requiring redressing, than either of the others.

Other common metals will suggest themselves. Plugs of several metals or alloys, and similar plugs of liner steel for comparison, could be screwed into prepared holes extending part way through that portion of the steel liner where the erosion is greatest. After firing, examination would show any real differences between the plugs, and such results should be confirmed by a liner made of the material showing the least erosion.

It may be possible to deposit electrolytically a heavy shell of nickel, copper, bronze, brass or even iron on the inside of the tube, and then bore out and cut the rifling in this shell. My belief is that any pronounced relief will not be obtained by a study of the effects of erosion, but by actual trials of the more common metals as liners. These tests must be made in the bore of a gun, not in bombs, the breech block, or in the powder chamber.

W. M. Corse,* Niagara Falls, N. Y. (written discussion†).—In Mr. Unger's very practical discussion of this subject he mentions the fact that aluminum bronze has been tried in small arms, where it offered good re-

* Manager, Titanium Bronze Co., Inc.

† Received Mar. 21, 1918.

sistance to erosion but failed on account of lack of strength. In this connection it might be of interest to note that a 10-per cent. aluminum bronze properly heat-treated will show the following physical properties:

| | |
|---|---|
| Ultimate tensile strength, lb. per sq. in.. | 85,000 to 100,000 |
| Yield point, lb. per sq. in.. | 43,000 to 50,000 |
| Elongation in 2 in., per cent.. | 12 to 18 |

The values for this metal are very close to those which Mr. Unger states as necessary for the steel for gun liners. It would seem possible, therefore, to adopt an alloy like aluminum bronze for such service if it showed superior resistance to erosion.

Aluminum bronze has a relatively high conductivity, can be as easily machined as steel of about the same properties, it resists abrasion remarkably well, and is fairly hard.

From the standpoint of cost, the scrap value of a liner made from bronze is so high that its ultimate cost might not exceed that of a steel forging.

C. I. B. Henning,* Wilmington, Del. (written discussion).—I have read Prof. Howe's paper with great interest and have nothing to add nor any question to raise considering the subject from the metallurgist's point of view.

The possible palliatives of erosion, as discussed by Prof. Howe in paragraph 57, are quite different if the minor calibers of ordnance are considered. This is especially true in the case of small arms. While no very great advance has been made recently in reducing the erosion experienced in major-caliber ordnance by changes in the propellant or in the detail of loading, it will be of interest to know that very great improvements have been made, and that further improvements can be effected as to erosion in small arms, by suitable changes in the propellant. Considerable diminution of erosion in field guns has been gained by the use of a cardboard wad at the base of the projectile, which by its combustion serves to cool the gases escaping past the base of the projectile in the early part of its motion in the bore of the gun. Experiments have been made with special cooling compounds placed at the base of the projectile for larger-caliber guns, but, so far as I am aware, the results have not been sufficiently promising to warrant general adoption. The principal of cooling the gases at the base of the projectile apparently finds its most useful application to the smaller-caliber guns only. As a special illustration of this, it may be interesting to know that in the case of the Mark VII .303 British service ammunition, using cordite as the propellant, a cardboard wad is loaded in the cartridge case at the base of the bullet. This addition has a very marked effect in reducing the erosion that is experienced with cordite when this small cardboard wad is omitted.

* Ballistic engineer, Experimental Station, E. I. DuPont de Nemours & Co.

In the case of small arms, the most satisfactory and effective way of reducing erosion is by the use of a propellant which burns at the lowest temperature consistent with obtaining the desired ballistic with a satisfactory weight of charge. Nitrocellulose powders of the "progressive burning" type, containing cooling agents on their surface, have been found to reduce erosion, and increase the accuracy life of a gun by two or three times. Powders of this latter type are not practical for major-caliber ordnance, because with a normal powder the pressure at the muzzle is already all that the gun can be expected to stand. In the larger-caliber ordnance we cannot reduce the pressure at the breech and yet maintain the required muzzle velocity, as we can in small arms.

James E. Howard,* Washington, D. C. (written discussion).—The writer desires to add a few thoughts acquired during the examination of the bores of guns.

The effects of powder gases on the metal at the bores of guns, to which the general term erosion has been applied, was made the subject of inquiry during the early period in the construction of modern steel ordnance.

The temperature of combustion of gun powder is very high; and prolonged exposure to such temperatures would melt the gun. The interval of time, brief as it is, at each time of firing, exposes the surface of the bore to a wash heat which has an effect on the steel. The duration of exposure to this high temperature is a maximum at the breech end of the bore, and a minimum at the muzzle end of the gun, the difference being represented by the time required for the projectile to travel the length of the bore. This difference in the length of time of exposure is marked by a difference in effect on the steel at the surface of the bore and the metal next below. There is a progressive difference in the character of the surface of the bore and the state of internal strains set up in the metal in that vicinity in passing from the powder chamber of the gun to its muzzle, showing a state of maximum severity at the breech end.

In the bores of guns in which fixed ammunition is used, the effects of the powder gases are first felt at the forcing cone and the beginning of the rifling. There is scant or no evidence of the escape of the gases past the bands of the projectiles, which immediately engage with the rifling. In small arms, however, guttering takes place just in front of the powder chamber after endurance firing.

The surface of the powder chamber is protected by the cartridge case, and in small arms and guns using fixed ammunition, little occurs through violent thermal changes to disturb the metal in this part of the gun. All parts of the bore, however, from the forcing cone to the muzzle are directly exposed to the action of the powder gases.

* Engineer-physicist, Interstate Commerce Commission.

The effect of rise in temperature, independent of interior pressure in the gun, is to cause over-compression of an annular zone of metal at the surface of the bore. This state of over-compression is momentary, and by reason of the rapid cooling of this intensely heated zone is followed by a reversal of the state of internal strain from compression to that of tension. The rate of cooling of the heated annular zone of steel, by conductivity of the metal of the tube beyond, is sufficiently rapid to result in the hardening of a thin layer of the metal. Internal strains of tension are acquired of such magnitude that the surface of the bore is broken up by a network of fine cracks.

These thermal cracks undoubtedly open and close according to the relative temperature of the surface metal and that of the interior of the walls of the gun. Successive waves of heated metal pass into the walls during the period of firing the gun, a limited rise in temperature eventually reaching the exterior surface in guns of large caliber. A temperature uncomfortable to the touch is soon acquired by small arms. Owing to variations in temperature in the walls of guns of large caliber, the stresses in the successive annular layers of the tubes, jackets, and hoops will necessarily vary from time to time. The first round fired from a cold gun would seem to be a critical one, in a certain respect, at least in that part of the tube immediately in advance of the band of the projectile, as the latter travels along the bore, since the rate of transmission of the strains due to the interior pressures of the gases would be greater than the acquisition and transmission of heat waves.

To endure these thermal changes, however, is not the principal function of the gun. It must sustain an interior pressure of 40,000 lb. per square inch, more or less. This interior pressure causes a greater tangential than longitudinal stress in the walls of the gun. In the course of time the thermal cracks penetrate more deeply into the metal and under the influence of the tangential stresses naturally develop more rapidly in longitudinal direction, than transversely.

In addition to the interior bursting pressures, local strains are brought upon the forcing sides of the lands of the rifling, the strains required to impart rotary motion to the projectile in its flight. The soft metal of the band of the projectile, usually copper, is forced into the cracks which are formed in the angles of the grooves and lands of the rifling, not infrequently obscuring the presence of thermal cracks which may have considerable width and depth of penetration.

The presence of longitudinal cracks is a menace to the integrity of a single-forging gun, and such cracks may also be a menace to a built-up gun. The strains in the walls of a gun during firing, as it is well known, are at a maximum at the surface of the bore, decreasing very rapidly in the superimposed metal. When a gun bursts, not due to failure of the

breech mechanism, the fracture may be expected to begin at the surface of the bore.

It is recognized, in the present design of guns, that the metal at the surface of the bore will be injured when a limited number of rounds have been fired, while all other parts of the gun remain unaffected, and liners are accordingly introduced within the tubes. The function of the liner is to receive and transmit the interior pressures to the tube and carry the rifling. No account is made of the strength of the liner, which is expected to be renewed after the condition of the bore is impaired. Liners were used in British guns from an early date, and are now adopted in the guns of the United States.

Concerning erosion of the bore from rush of heated gases along the tube, whatever the velocity of discharge at the muzzle may be, there can hardly be such action at the bottom of the bore, where the heat effects are greatest. The corners of pressure-gage plugs, when located in the powder chamber, are rounded by exposure to the high temperature of the combustion of the powder charge. Thermal cracks are formed on the exposed surfaces of the plugs.

Apparently the phenomena observed in the state of the metal, on surfaces exposed to powder gases, are explained by the raising of the surface of the steel almost to the melting point, followed by refrigeration due to the conductivity of the mass of the metal next beyond. The bores of guns seem to afford unique examples of combined sudden heating and cooling, attended by strains resulting from the generation of great interior pressures.

Francis I. DuPont, Wilmington, Del. (written discussion).—The problem of lessening erosion in guns, as I see it, does not stand by itself, but is intimately connected with and primarily dependent upon the design of guns and projectiles, as well as upon the character of the powder. These, in turn, are dependent upon military requirements, about which the civilian has only a secondary knowledge.

I will refer to two statements on page 516 of Prof. Howe's paper: "Erosion must needs increase rapidly with the size of the gun and the weight and velocity of the projectile." "The heating effect must increase with the length of the gun."

While these statements are true, they do not seem to me to be all of the truth. I would state it this way: The heating effect, also erosion, varies directly as some function of the weight of the projectile per square inch of base area. While I do not go so far as to say that the size and length of guns have nothing to do with the matter, I wish to call attention to the fact that it is in reality the weight of projectile per unit of base area which is by far the greatest factor.

For years I have been confronted with the problem of making powder

which lessens erosion, and it was the apparent hopelessness of accomplishing very much in this way that led me to scrutinize to some extent the basic premises from which gun designers have arrived at their present conclusions.

In support of the proposition that heating effect and erosion depend upon the weight of projectile per unit of base area, I would invite consideration of the probable effect of firing a gun 1 in. in diameter and 50 ft. long with a projectile about the length of the 12-in. 50-caliber projectile, weighing the same per unit of area as the 12-in. projectile. Might we not expect to get erosion comparable with that noticed in a 12-in. 50-caliber gun?

Is it not in reality the time required for the pressure on each square inch of the projectile base to start and accelerate the mass of metal, etc., ahead of that square inch of area which alone allows the heating effect to take place.

As to the length of the gun: Since most of the time is required for the first few feet of travel of the projectile, it is clear that shortening the gun would lessen this time factor only according to a gradient which is not steep until the gun is very much shortened.

These considerations lead the gun designer and the chemist immediately outside their domain, and compel them to ask of the military expert whether a lighter projectile fired with a higher velocity would be more or less effective than the present heavy projectile fired at the present velocities.

I realize that the question of erosion is less important than the proper weight of projectile for military purposes, but I wish to point out that a lighter projectile fired at a higher velocity would result in very much less erosion in our heavy guns; also that, in small arms, the 150-grain bullet, travelling at 2700 ft. per second, has been substituted for the 220-grain bullet travelling at 2000 ft. per second, and that this has been done in spite of arguments in favor of the heavier bullet which seemed convincing.

Zay Jeffries,[1] Cleveland, Ohio (written discussion).—Prof. Howe's explanation of the martensitization, Secs. 6 to 26 incl., is so clear that one can see clearly with his mind's eye the sub-microscopic particles of the sorbite merge to form austenite and then change to martensite on cooling.

That the hardened layer is martensite there seems to be little question. It is not, however, the usual type of martensite encountered in heat-treated steel. The needle structure usually found in ordinary martensite may be considered as a residual effect of the cleavage or gliding planes in the austenite from which the martensite was formed on quenching.

[1] Director of Research, Aluminum Castings Co.

The larger the austenite grains, the more pronounced, therefore, will be the needle structure in the resulting martensite. The conditions existing on the heated surface of a gun bore are not conducive to the formation of large austenite grains. In the first place, the ferrite and cementite which have not completely merged into austenite will act as mechanical obstructions to the grain growth of the merged or equalized austenite. Again, the time available for the coalescence of the grains of equalized austenite is so short that it is doubtful whether grains of appreciable size could form. The rapid heating of the steel through the *Ac* points would also tend to produce exceedingly small grains of austenite. This is especially true when it is considered that the sorbite from which the austenite must be formed consists of particles of cementite and ferrite, some of which, at least, are so small that they cannot be seen with a high-power microscope. The tendency under these conditions would be to produce very small grains of austenite, probably sub-microscopic. On quenching this very fine-grained austenite one would not expect the resulting martensite to have a needle structure, since the needles, if they are present, would also be sub-microscopic in size. The face of the bore would be more likely to have martensitic needles developed than any other part of the liner, and a thin layer of the bore face is eroded away each round.

This explanation of the lack of martensite markings is not exactly in accord with Prof. Howe's idea as stated in Sec. 18. He implies that the typical needle or acicular martensite structure might form if time were given during the quenching period. In other words, he implies that this structure is due to the particular type of crystallization of martensite, whereas it seems to me that the needle structure is not at all typical of martensite, but, as above mentioned, that it is due entirely to the cleavage or gliding planes of the austenite. It would not be surprising if austenite grains of sufficient size formed at times in a gun, to produce typical martensite.

The martensite at the surface of the gun bore is harder than ordinary martensite of the same carbon content. This excess hardness is in keeping with the general proposition that martensite of a given carbon content is harder when produced from small austenite than from large austenite grains. If, in this case, the austenite grains are sub-microscopic in size—a condition which we cannot obtain ordinarily in practice—we might expect that the martensite produced from such austenite would be harder than martensite with the same carbon content produced during an ordinary heat-treatment process.

Hardness tests on certain sections of 12-in. gun liners have been made, with results given in Table 1. The increased hardness on the bore face is, of course, that due to the hardened layer having the thickness stated in the note to Table 1. Since the deformation caused by the scleroscope

was less than that of the Brinell ball, the former shows a greater increase in hardness of the bore face over that of the steel in its original condition. A scratch test on the hardened surface shows a still greater increase above that of the steel in its unchanged condition.

TABLE 1.—*Hardness Measurements on 12-in. Gun Liners*

| Steel No. | Location of Section | Brinell Hardness, 1000-kg. Pres. | Scleroscope Hardness | Remarks |
|---|---|---|---|---|
| 1 | Origin of rifling.... | 161 | 25–28 | Halfway between bore face and outer wall. |
| 1 | Origin of rifling.... | 218 | 45 | Bore face on land. |
| 1 | Origin of rifling.... | 200 | 45 | Bore face in groove. |
| 1 | 228 in. from muzzle | 164 | 28 | Halfway between bore face and outer wall. |
| 1 | 228 in. from muzzle | 200 | ....... | Bore face on land. |
| 2 | Origin of rifling.... | 171 | 26 | Halfway between bore face and outer wall. |
| 2 | Origin of rifling.... | 200 | 35 | Bore face on land. |
| 2 | Origin of rifling.... | 185 | ....... | Bore face in groove. |

NOTE.—Analysis of steel No. 1: C, 0.474; P, 0.026; Mn, 0.699; Si, 0.193; S, 0.022. Martensite layer 0.0063 in. thick at origin of rifling; average of 0.002 in. on land 228 in. from muzzle. Martensite layer 0.008 in. thick on driving edge of land 228 in. from muzzle.

Analysis of steel No. 2: C, 0.428; P, 0.029; Mn, 0 633; Si, 0.228; S, 0.022. Martensite layer at origin of rifling 0.005 in. thick.


FIG. *A*.—Two types of erosion in 12-in. gun liner at origin of rifling, after useful life. × 2.

Although these steels are similar in chemical analysis and in structure (steel No. 2 has slightly less excess ferrite than No. 1) their bore faces at the origin of rifling, after about the same amount of erosion, are entirely different in character. These are shown in Fig. *A*, magnified 2 diameters. Steel No. 1 has very few cracks and these run longitudinally both on land and groove. Aside from these few cracks the surface is bright and smooth. Steel No. 2 has its bore face covered with a network of deep cracks. Steel No. 1 corresponds to Prof. Fay's[2] Fig. 1-*E;* while No. 2 corresponds to Prof. Howe's Fig. 17. Are these the only two common types, or are there several?

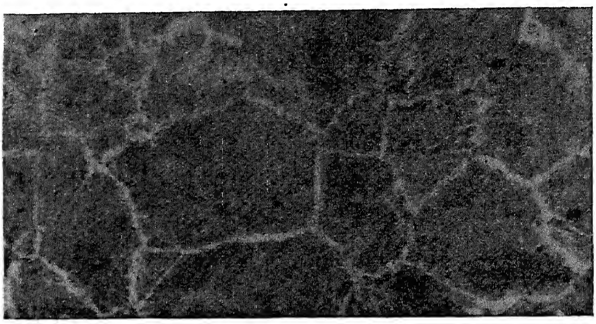

FIG. *B.*—Bore face of No. 1, Fig. *A*, showing ferrite network extending into martensite. × 100.

Fig. *B* is a micrograph, 100 ×, of steel No. 1 at the origin of rifling, showing the layer of martensite into which the ferrite network extends.[3] Certain portions enclosed within one network of ferrite can be recognized as part martensite and part sorbite. This condition accords with Prof. Howe's statements about the time necessary for the merging. The ferrite which now extends into the martensite gradually decreases in quantity toward the bore face, as would be expected. This steel favored the formation of martensite, because the sorbite contained nearly 0.9 per cent. carbon. It did not favor the transformation of all the steel into martensite of 0.47 per cent. C (the carbon content of the steel) because the excess ferrite was in relatively large masses and hence required more time than was available in which to dissolve in the higher-carbon austenite.

One fact of significance is to be gained from Fig. *B*. The flow of the metal 0.001 in. and more from the face of the bore, has been negligible. A longitudinal section of this steel was examined at the origin of rifling,

[2] *Trans.* (1916), **56**, 468–482.

[3] It will be noted that the martensite etches darker than the sorbite. The reverse was true after slight etching, but heavy etching makes the martensite look like troostite with slight etching.

and no distortion of the ferrite network could be seen in the direction of the gas flow. In some places the ferrite network extends to within 0.001 in. of the bore face. Of course, the deformation due to thermal expansion, which Mr. Maxim[4] refers to as "displacing itself upon itself," would not be detected in this manner.

*The Probable Influence of Pressure.*—Prof. Howe has not considered in detail the influence of pressure of the gases back of the projectile. Prof. Fay considers that the high pressures developed might aid in the formation of the hardened layer. He quotes Spring[5] as authority for the statement that brass could be produced at ordinary temperature by subjecting powdered zinc and copper to enormous pressures. The work of Spring along these lines has largely been discredited by later investigations. Johnston and Adams[6] review Spring's work, together with that of later investigators, and the conclusion is reached on considerable evidence that the effect of pressure on hastening the solution of one component in another at temperatures lower than these components ordinarily dissolve in each other, is very slight and may be either helpful or harmful. Hallock,[7] Rosenhain and Tucker,[8] and Masing[9] all show, with the aid of the microscope, that Spring was in error regarding the effect of pressure on the solution of one substance in another. High pressure would, however, tend to promote the $Ac_1$ transformation because a reduction in volume accompanies it.[10] The increased pressure would also tend to decrease the mobility of the molecules and might actually retard the speed of the transformation. Johnston and Adams state that 15,000 atmospheres uniform pressure will not change calcite (2.71 sp. gr.), to arragonite (2.93 sp. gr.), nor marcasite (4.9 sp. gr.) to pyrite (5 sp. gr.), at room temperature; nor will 2000 atmospheres pressure at 425° C. cause the latter transformation to take place to any marked extent, although it occurs at 450° at ordinary pressure. These examples, however, represent transformations in substances at temperatures of relatively low molecular mobility. If we consider the arbitrary boundary between the regions of low and high molecular mobility in a metal as the lowest recrystallization temperature after cold deformation, then the $Ac_1$ point in carbon steel is in a region of high molecular mobility. Al-

[4] *Trans.* (1916), **56**, 486.

[5] W. Spring: *Berichte der deutschen chemischen gesellschaft* (January–June, 1882), 595.

[6] John Johnston and L. H. Adams: *American Journal of Science*, Ser. 4 (1913), **35**, 205–253.

[7] William Hallock: *American Journal of Science*, Ser. 3 (1889), **37**, 402–406.

[8] W. Rosenhain and P. A. Tucker: *Philosophical Transactions of the Royal Society of London*, Ser. A (1909), **209**, 120.

[9] G. Masing: *Zeitschrift for anorganische chemie* (1909), **62**, 265–310.

[10] A. Sauveur: *Metallography of Iron and Steel* (1912), 2, Lesson 9.

lotropic modifications, in regions of high molecular mobility, should behave, under pressure, in the same manner as sulphur and phosphorus at high pressures and relatively low temperatures. High pressure tends to promote the formation of the modification having the greater specific gravity. The magnitude of the lowering of $Ac_1$ at the pressures encountered in the gun, say 2000 atmospheres, should be very slight, probably in the neighborhood of 5° to 10°, and inasmuch as the speed of the transformation would be changed but little, if at all, by the pressure, we may conclude that the effect of pressure on the $Ac_1$ transformation is very slight under the conditions existing within a gun at the time of firing.

Some calculations on the change of melting point due to the increased pressure of the gun also indicate that this factor is so small as to be negligible. Johnston and Adams give the following equation for the determination of the change in melting point due to increase in uniform (hydrostatic) pressure:

$$T_2 = P_2 \left[\frac{T(Vl - Vs)}{41.3Q}\right]$$

The following values will be considered for a 0.4 per cent. C steel:

$T_2$ = change in melting point in °C.
$P_2$ = Pressure in atmospheres = 2000 approximately.
$T$ = Melting point in absolute °C. = 1700° approximately.
$Vl$ = Volume in c.c. of 1 gram of molten metal at melting point = 0.141 approximately.
$Vs$ = Volume in c.c. of 1 gram of solid metal at melting point = 0.130 approximately.
$Q$ = Latent heat of fusion in calories per gram = 66 approximately.
$T_2$ = + 14° C.

The value for the crystallization shrinkage at the freezing point is assumed as 9 per cent., which is the highest value I have been able to find even for cast iron.[11] I think 5 per cent. would be a safer figure for steel, and the increase in melting point would then be but 8°. The increase in melting point due to the pressure of the gases is about equal to that caused by a decrease in carbon content of 0.03 to 0.05 per cent. The time in which a pressure of 2000 atmospheres is maintained in a gun is only a small fraction of that in which the projectile is in the gun. We may consider, therefore, that the increased melting point due to gas pressure in that portion of the bore back of the projectile is so slight as to be of little moment.

There is another point, however, which may have greater bearing on erosion than the change in melting point. This is the increase in cohesion of the white-hot steel due to the pressure of the gases. The modulus of elasticity of the white-hot steel is very low, but the excessive pressure

[11] R. Moldenke: *Trans.* (1916), **56**, 455.

tends to force the molecules closer together than they would ordinarily be at the same temperature, and this increases the cohesion. At the instant the pressure is the highest, the temperature has probably not reached its maximum at the surface of the steel. The reduction in pressure is coincident with the flow of the gases, and at the moment the dynamic forces of the gases are the greatest it would seem that the cohesive forces in the steel begin to decrease, because part of the white-hardness is due to the pressure of the gases. The few degrees increase in the melting point of the steel, due to the pressure of the gases, would also favor liquefaction as the pressure was reduced, even though this action quantitatively should be slight.

*White Hardness and Grain Size.*—After weighing many factors, Prof. Howe attributes the greater part of the erosion to the action of the heated gases in motion. This idea is also held by many others. To resist the erosive action of the gases it is suggested that a metal should have the property of white hardness. In any given metal or alloy composed largely of one component, its white hardness will be the greater the larger its grain size.

I have shown[12] that in all single-component metals there is a temperature at which the cohesion, as measured by either the first permanent deformation or by the amount of deformation under a load somewhat above the elastic limit, is independent of grain size. It is true that this temperature increases as the time of application of the load decreases, but the normal position of this point, which I have called the equi-cohesive temperature (my interpretation being that it is the temperature at which the amorphous phase has the same cohesion as the crystalline phase) is about the same as the recrystallization temperature of the particular metal under consideration after severe cold work. In iron, for instance, the equi-cohesive temperature is about 550° C. Above this temperature, iron becomes softer as the grain size decreases; below it, iron becomes harder as the grain size decreases. Any metal in the temperature region of grain growth in the solid state will be harder as the grains become larger. The difference in grain size makes but slight difference in hardness near the equi-cohesive temperature; but as the melting point of the metal is approached, the fine-grained material is very much softer or less cohesive than the coarse-grained material.

It does not seem possible to heat any of the steels now used for gun liners to temperatures near their melting points without at the same time producing very fine-grained structures. For example, the austenite grains may be considered to be very small—perhaps sub-microscopic—under the firing conditions. With a very low-carbon steel the ferrite, on changing to gamma iron, might also be expected, with the very rapid

[12] *Journal of the American Institute of Metals* (December, 1917), **11**, 300–324.

heating conditions, to change into very small grains of gamma iron. The same would be true of a liner of pure iron. The successive rapid heatings should produce very small grains of gamma iron when the metal is heated to a temperature even near melting. Inasmuch as small grains, other conditions being equal, should reduce the white-hardness of the iron or steel, might we not expect greater resistance to erosion in a material in which fairly large grains were stable even up to the melting point of the metals? To meet the conditions of high melting point and high heat conductivity, and yet maintain a large grain size at high temperatures, seems very difficult. Silicon-steel has a rather high melting point, although not so high as carbon steel. It can be produced, however, with a large grain size and these grains are stable up to the melting point.[13] It would be interesting to test some silicon-steel erosion plugs with varying grain size. Even if this material does not prove as resistant to erosion as the steels now in use, the information regarding change in grain size would be definite and possibly valuable.

The results of Admiral Earle[14] with annealed steel would indicate that pure iron resists erosion at the high temperatures better than austenite containing considerable carbon. Even though the gamma iron would have a small grain size due to the extremely short time available for growth to take place, it would still have a grain size much greater than that of austenite containing considerable carbon and consequently it would contain less amorphous iron. If some way could be devised to keep the ferrite grain size larger, I should think that the tendency would be to reduce the erosion. Besides increasing the white hardness, the coarse-grained material would offer more resistance to melting than fine-grained metal.

Henry Fay,* Cambridge, Mass. (written discussion).—One year ago I presented my paper[15] on the "Erosion of Guns" with the hope of bringing out some discussion and getting the view of others on this complex and important problem. I had hoped that Prof. Howe might discuss the paper at the time, but I am more than gratified that he should give it the attention which he does now. He has turned upon the problem his clear and logical mind and has greatly illuminated the subject, and I sincerely hope that others will contribute their share toward clearing up the problem.

I wish to discuss at this time just two phases of Prof. Howe's paper: first, the effect of pressure in promoting martensitization; second, oxidation as a cause of erosion.

In my study of the problem I have examined many fragments of

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[13] W. E. Ruder: *Trans.* (1913), **47**, 569–583.

[14] *Trans.* (1916), **56**, 491.

* Professor of Analytical Chemistry, Massachusetts Institute of Technology.

[15] *Trans.* (1916), **56**, 468.

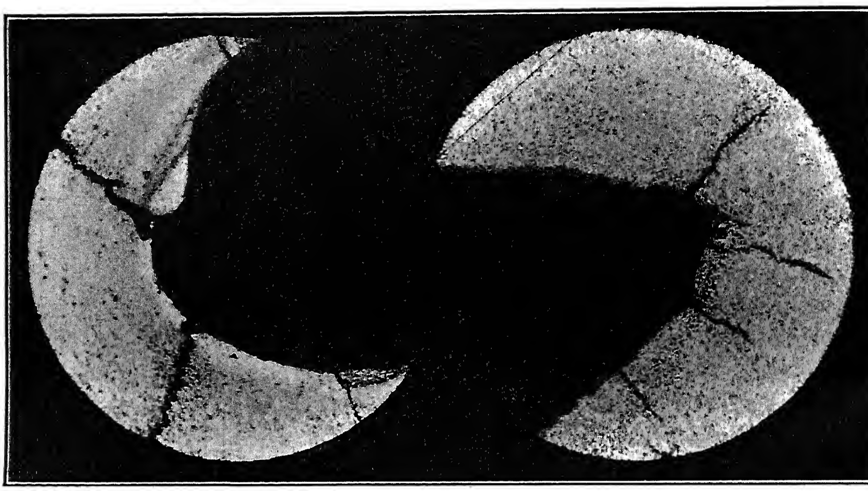
many guns, and I offered as a working hypothesis the idea that where the maximum pressure occurred, there also took place the most rapid hardening of the surface. Having eliminated cementation and cold work as being the primary or sole cause and having established beyond doubt that the hard layer was martensitic, in spite of the fact that the needle structure was rarely present, it seemed to me evident that there was a contributory cause, especially so since tubes through which gases had been allowed to escape, but which had not been worked by a projectile, were always clearly and unmistakably martensitic.

Prof. Howe objects to the idea advanced by me and explains the differential hardening on some of my pressure plug letters as being due to salience, or gas currents; in other words, what cannot be explained by salience is explained by gas currents. In pointing out that salience is an important factor, Prof. Howe has made a distinct contribution, and I am glad to accept it as a prime factor in hardening. In regard to gas currents being a factor in hardening, I am much in doubt and I cannot see how it can be possible to explain some of the facts on this hypothesis. In fact, it is difficult to conceive of differential gas currents in a space which is under such great pressure, and where it is necessary to assume the gas to be moving in opposite directions on surfaces less than 1/4 in. apart. To take a specific example, let us examine, the letter S stamped upon a pressure plug. The steel hardens in the lobes of the letter. Prof. Howe states that there is difficulty in telling where the maximum pressure occurs, but in this case there can be little doubt that it occurs in the two lobes. Salience may be a factor in this differential hardening, as in stamping the letter the metal is pushed outward into a salient and to some extent upward from the surface. This effect, however, is diminished by the polishing in preparation for microscopic examination which was made before having placed the plug in service. All of the upward salience was removed by this preparation, and in spite of the salience it is difficult to see why one side of the letter should harden more than the side directly opposite, unless one side is worked more than the other. The worked side does not project into space one bit more than its neighboring side. If it projected further one would certainly expect it to absorb more heat, but this is not the case. It is impossible to say positively that salience is not the prime cause. On the other hand, the assumption that work is a factor not only explains this case satisfactorily but also many other cases where salience does not seem to be a factor.

In order to explain the phenomena by the action of gas currents, it would be necessary to assume currents moving in opposite directions against the upper and lower loops only 1/32 in. apart. I am fully aware that this does not disprove salience and gas currents as factors, and does not prove pressure as a positive factor.

Again, I submit photographs of the period stamped at an angle of 45°, which show that hardening took place at the obtuse instead of the acute angle. Certainly the point of salience is at the acute angle in this case, but the point of maximum pressure is at the obtuse angle undoubtedly. This has taken place after the removal of some of the surface by polishing, as shown in my Fig. 9[16] and is practically the experiment recommended by Prof. Howe in paragraph 21, although not carried to the extent recommended by him. I regret very much that I am unable at this time to report upon the pressure plug prepared at his suggestion.

In regard to Prof. Howe's contention concerning my Fig. 6, he assumes



that it can be explained by gas currents, and does not see why the right-hand side of the male apex and the left-hand side of the female apex are hardened. If the die had been held at a slight angle, plastic deformation would have taken place exactly as indicated in his Fig. 21, and would give a similar effect to the obtuse angle in my period stamped at an angle of 45°, hardening taking place on the obtuse angle in this letter. In this case one would think that if salience were the most important factor it would be exercised at the point of the male apex of the letter A, and that if gas currents were the important factor the whole of the apex and not one side would be hardened.

In regard to Prof. Howe's paragraph 42, I wish to quote from a report submitted to the Commanding Officer, Watertown Arsenal, November, 1913: "Having a surface heavily fissured from heat cracks, the escape of the gases at high velocity around the rotating band through these

[16] *Op. cit.*, 477.

cracks and the rush of the gases back of the projectile soon produce an erosion of the surface. The conditions existing in the explosion chamber and for some calibres in front are very complicated, and it is difficult to say just what these conditions are at any moment as they are dependent primarily upon the nature of the explosive, the temperature of combustion, and the pressure of the gases.

"The principal products of the reaction are carbon monoxide, carbon dioxide and water vapor. There will be for any particular temperature an equilibrium between hydrogen and oxygen expressed by the equation:[17]

$$2H_2 + O_2 \rightleftarrows 2H_2O$$

in which the direction in which the reaction takes place is dependent upon (*a*) the concentration of the reacting masses, (*b*) the temperature, and (*c*) the pressure. With high temperatures the reaction will go largely from right to left, and if so there would be further reaction between the iron and oxygen producing magnetic oxide, $3Fe + 2O_2 \rightleftarrows Fe_3O_4$. This in turn would react with hydrogen:

$$Fe_3O_4 + 4H \rightleftarrows 3Fe + 4H_2O.$$

The conditions thus involving hydrogen, oxygen, water vapor and iron are complicated, but the equilibrium phenomena involving carbon monoxide, carbon dioxide, iron, iron oxide, and iron carbide are even more so. The result is that it is highly probable that a certain amount of oxidation takes place and that some of the oxide formed is swept away by the rush of gases. The greatest sweeping action would take place where there is the greatest velocity of gas, and that is where the gas escapes past the rotating band through the heat cracks. The subsequent slower rush of the large mass of gas would sweep the surface clean and probably develop new heat cracks. The escape of gas past the projectile takes place to the greatest extent in the primary stage of the explosion, that is, before the projectile has attained its maximum momentum. Equilibrium is soon reached, however, and the gas and projectile move forward at the same rate. From the point where this equilibrium is attained forward to the muzzle there should be less violent erosive action, although the maximum erosive action will take place at the point where there is the greatest number of heat cracks.

"The greatest erosive effect will be produced by that powder which is most strongly oxidizing in character, assuming similar conditions of temperature and pressure. There may be some hope for relief in the introduction into the powder of an inert substance which will either be strongly reducing in itself or will influence the direction of the reaction

[17] The reaction $2CO + O_2 \rightleftarrows 2CO_2$ is just as important.

toward the non-oxidizing products. Ozokerite[18] probably helped by reason of the carbon and hydrogen helping to decrease the oxidation of the metal. Any product liberating much hydrogen would have the maximum effect in reducing erosion."

It will be seen that at that time I was strongly impressed with the idea that oxidation was an important factor, and I am convinced that this will be verified when someone has established the equilibrium data along the length of the rifle.

F. N. Speller,* Pittsburgh, Pa. (written discussion†).—The conditions involved in the wear of the interior of guns seem very different from those surrounding the pipe balls used in the lap-welding of pipe, referred to on page 566, except in respect to the life of each, which amounts to only a few seconds of actual use. Good cast-iron welding balls are better than any steel balls except manganese steel, so that in this case the lack of welding quality in the ball seems to determine its life rather than the temperature of the surface or melting point of the ball. The temperature of the skelp is about 1400° C. which, with the high pressure of the rolls, tends to weld the skelp to the ball.

The main factors involved in the wear of gun liners seems to be:

1. The hardening of the inner surface, which is shattered and cracked by the impact of the next charge and sudden expansion of the metal. The stresses are applied so suddenly as to be of the nature of impact, so that the inner machined surface should be so made as best to withstand this character of stress.

2. The erosive sweep of hot gases carrying partially burned particles of powder.

3. The softening of the inner surface of the liner is undoubtedly an important factor, although there does not appear to be sufficient evidence to prove that the surface of the liner is actually melted.

The first and last of these conditions might be partly met in practice by means of a steel liner with a higher melting point, which would not harden and would therefore be less susceptible to cracking and wear; but whether any such means would pay would have to be determined.

Henry R. Batcheller, Washington, N. H. (written discussion).—Ten years ago it would not have been considered possible to determine molybdenum in steels of the character described in paragraph 58, containing so large an amount of nickel and chromium. An analysis of ferro-molybdenum made about 1907 by commercial methods gave what

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[18] Referring to some experiments which had been made using this substance. Of course, the introduction of any inert substance will diminish the muzzle velocity.

* Metallurgical Engineer, National Tube Co.

† Received Feb. 25, 1918.

purported to be the following results: iron, 92.55; carbon, 5.64; molybdenum, trace.

It subsequently developed that the "carbon" was mostly molybdenum. The consulting chemist was C. T. Hennig, a former employee of Krupp, who represented them in this country in certain technical ways with special relation to molybdenum, chromium and manganese. The firm producing this ferro-molybdenum was offered a contract for the sale of its whole production to Krupp, who used it for ordnance purposes and would have been glad to get more. According to Hennig, the physical properties of the steel were improved by the addition of molybdenum just before pouring; he also said that analysis would often show only a trace of molybdenum in the steel, while the slag would be high and some molybdenum would be volatilized, especially if the molybdenum were added in the form of metal and not as ferro-molybdenum. When using the metal, it was, therefore, necessary to protect it from oxidation by plating, and silver plating was used successfully. This information may be of service to steel makers at the present time.

W. P. Barba,* Washington, D. C. (written discussion).—Referring to Prof. Howe's suggested use of special alloy steels for gun liners, I wish to emphasize the economy, as well as the metallurgical advantage, of using manganese alloys in the molten state. A highly undesirable practice has crept into many steel works; that is, to recarburize in the ladle almost wholly, adding large volumes of coal and of broken ferromanganese, the melters hoping that, by Divine Providence, or some agency outside themselves, this carbon and manganese will be thoroughly melted and evenly diffused throughout the whole mass in the ladle.

This is rarely the case, and the last one, two, or three ingots in the line are frequently found with as much as 20 points variation of manganese, and 10 points variation of carbon, due to the late melting and uneven diffusion of the recarburizer. Shell steel, or steel to be heat-treated, produced by such a poor method of adding recarburizer, is so heterogeneous in composition that uniform heat treatment cannot be applied with uniform results in the product.

Methods of melting these alloys should be carefully considered, as cupola melting results in possibly even greater loss than if the manganese were added under the slag in the furnace.

The rapid advance in the use of electric melting furnaces has made it very easy to melt properly and with slight loss. In many places the power cost is sufficiently low to offer real economy by electric melting of a ferro-alloy of such high intrinsic value as our manganese alloy at present. After melting, the alloy should be added by pouring from the tilting furnaces into a ladle hanging on a crane having a weighing scale in the system.

* Lieutenant-Colonel, Ordnance Department, U. S. A.

Tho. H. Poulsson,* Norway.—We do not possess much experience in the use of low-manganese alloys at our own works, because we specialize in high-grade steels, so we must use a good alloy, generally 60 to 80 per cent. manganese.

Spiegel-eisen (10 to 12, or, better, even 8 to 10 per cent. Mm., according to the prescribed analysis) can replace ferro-manganese entirely in all grades of steel as used for rails. The spiegel should be melted in an electric furnace, on account of both the manganese and the carbon—and not in cupolas. The necessary addition of silicon must be given in the form of ferro-silicon, of which you have doubtless sufficient in America.

While steel of rail quality already consumes, for its production, large quantities of manganese, the production of mild steel (which must in your country be a factor of far greater importance, when considering the ordinary sections, steel plates and angles, etc.) absorbs far greater quantities of manganese, which, in steels of this quality, can be replaced by calcium carbide.

The following illustrates what could probably be saved: Assuming that America has to make 10,000,000 tons of soft steel, this would ordinarily require 50,000 to 60,000 tons of 80-per cent. ferro-manganese, which can be reduced to 20,000 to 30,000 tons of 60-per cent. ferro-manganese by using calcium carbide. The effective influence of the carbide depends largely on the furnace, whether fixed or tilting. The essential point in using carbide is that the pure steel shall hit the carbide in the ladle, before the slag comes in; therefore, with tilting furnaces, in which the slag can be held back, the result is good and the effect is uniform.

The following figures are not to be looked upon as accurate, but should be fairly approximate: Per ton of steel, about 2 to 3 kg. of 60-per cent. ferro-manganese (which should be added in the furnace), and about 7 to 8 kg. of calcium carbide (diameter of grains 7 to 10 mm.). The deoxydation, more or less active, should be assisted by adding finely pulverized 50-per cent. ferro-silicon, or aluminum. The exact quanity of carbide to be used is very easy to learn by experiment.

We do not believe that silico-manganese can be used for making soft steels, but can, of course, be used to great advantage when making higher silicon steels.

E. D. Campbell,† Ann Arbor, Mich. (written discussion‡).—Prof. Howe's careful metallographic study of rings cut from the lining of a 14-in. naval gun throws a great deal of light on the probable mechanism

* Director of The Stavanger Steel Co.

† Director of Chemical Laboratory, University of Michigan.

‡ Received Mar. 4, 1918.

of much of the erosion which results from the firing of the guns. His results demonstrate that the erosion on the lands is considerably greater than in the grooves, although the cracks in the latter seem to be nearly as completely filled with copper derived from the driving band as are those in the lands, on which the pressure of the driving band must be considerably greater than it is in the grooves.

In the experimental work carried on at the Watertown Arsenal, the results of which were described in Prof. Fay's recent paper,[1] the formation of a hard surface is clearly due to what Prof. Howe has termed "martensitization" of the bore surface. Prof. Howe, calling to mind the gradual solution of a crystal of salt immersed in water, attributes the "martensitization" of the bore surface to the gradual, progressive merging of the cementite with the ferrite until mergence of these two components is complete in the extreme bore surface, and the rapid cooling by conductivity converts this solid solution into martensite.

It would be interesting to know whether Prof. Howe uses the term "cementite" in a generic sense, as it is now used by many metallurgical chemists, that is, as indicating a metallographic constituent of steel consisting chemically of a more or less complex mixture of carbides, rather than a single carbide, just as the term "petroleum" is used for a large number of oils consisting of mixtures of different hydrocarbons.

Recognizing the unity of mechanism of metallic and of aqueous solutions, might it not be a strictly correct interpretation of "martensitization" to say that this was due to the gradual, progressive solution of the carbides in the iron, and that these remained in solid solution because the bore surface was cooled so rapidly by conductants? In addition to the expansion of the bore surface due to temperature alone, would it not be probable that the osmotic pressure of the dissolved carbides would add very decidedly to this expansion, thus increasing the number and size of the cracks which would develop on sudden cooling?

It is, of course, common knowledge that, if a file is hardened by very sudden cooling in water, and not tempered at all, the teeth will be very easily stripped, even by filing a soft body like copper, especially if the file strokes are very rapid. May not the increased mechanical erosion of the lands, as compared with the grooves, be due more to the mechanical stripping of the edges of the cracks by the rapidly driving band than to any other cause? For certainly the velocity of the gases and the temperature must be substantially the same on the lands as in the grooves.

It will, of course, be difficult to judge with any great degree of accuracy the proportion of the brittle bore surface which has been mechanically carried away by the driving band of the projectile, as compared with that portion which has been removed either by volatilization, oxidation

[1] *Trans.* (1916), **56**, 468.

or friction of the rapidly moving gases. The fact that, after the outward passage of each projectile the mechanical erosive action on the bore surface by the driving band is completely obliterated by the high temperature attained by the bore surface after the projectile has passed out, renders it impossible to secure a specimen which would show the effect of the driving band alone on the brittle bore surface.

Prof. Howe states that one of the important factors determining the temperature that would be attained by the bore surface during the explosion is the thermal conductivity of the steel. It would add value to the paper if Prof. Howe would give his ideas concerning the factors which determine the thermal conductivity of steel so that one might judge the extent to which the thermal conductivity could be practically controlled. Since a very considerable part of the erosion seems to be primarily due to the "martensitization" of the bore surface, would it not seem probable that if "martensitization" of the bore surface were reduced to a minimum, with a given temperature and pressure of the gases during the explosion, the erosion would be markedly reduced?

# Transverse Fissures in Steel Rails

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ON Aug. 25, 1911, a rail failed on the Lehigh Valley Railroad, causing a disastrous wreck. The surface of the fracture was in a plane at right angles to the length of the rail. There was a dark-colored, oval spot on this surface, located on the gage side of the head, representing the part of the rail which was the first to fracture. The nucleus of this spot showed that the fracture had an interior origin. Other fractures of this kind, the peripheries of which had not reached the surface of the rail, presented surfaces having a silvery luster. The darkened surface of the present fracture was caused, doubtless, by the air having had access to it.

This fracture was recognized as a fatigue fracture, of a modified type, its interior origin having been due to a component in the rail which is not usually present in cases of fatigue fractures. The metal next the running surface of the head of the rail was in a state of internal compression, this being the component not commonly met with in other cases, and which caused the fracture to have its interior origin. For the purpose of giving this modified type of fatigue fracture a specific name, for identification, the term "transverse fissure" was applied.

No mystery was thought to attach to its formation, or to its development. The explanation of its occurrence seemed obvious in the light of common knowledge upon the behavior of steel under repeated alternate stresses. By reason of the state of internal compression at the running surface, the maximum tensile stresses in the head, under bending loads, were along an interior element. Here the tensile strains were higher than in the fibers more remote from the neutral axis. Under such conditions the fracture of the rail would be expected to have an interior origin. A discussion of the subject of transverse fissures essentially becomes a discussion of fatigue fractures, since they are believed to be identical.

The descriptive term employed, merely for the identification of this type of fatigue fracture, had the appearance of creating alarm. Apprehensions arose that a new phenomenon in the physical properties of steel had presented itself, the explanation of which was sought in every

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detail in the manufacture of the rail, omitting consideration, however, of the effects of service conditions, where the explanation really lay. Under this illusory search the subject soon took on a nebulous state; the distinguishing characteristics of a transverse fissure were lost sight of, and quite dissimilar fractures were reported under the same general terminology.

In the classical tests of Woehler, some 50 years ago, we were told about the effects of repeated alternate stresses in causing the rupture of metals. Shreve, in 1876, in the preface of his translation of Spangenberg's experiments, said: "The Fatigue of Metals is the name which has been given to the effect produced by oft-repeated strains. * * * Spangenberg's experiments were in continuation of Woehler's. The results of these very important experiments have been before the profession for some years, * * * but tests of iron and steel still go on for the purpose of determining their elasticity, their elongation under strain, their ultimate strength and other qualities, while Woehler and Spangenberg's experiments show that it is very doubtful that these bear any proportion to the durability of the metals. These experiments prove that there is a limit of strain within which iron is practically indestructible, and that that limit is but little over 30,000 lb. per square inch for the best iron."

The Watertown Arsenal tests, 1888 to 1908, showed the limiting value for steels to be in the vicinity of 40,000 lb. per square inch (2800 kg. per square centimeter) for practically unlimited endurance of repeated alternate stresses. A certain number of repetitions of higher stresses, even as high as 60,000 lb. per square inch (4200 kg. per square centimeter) may be endured for a time by some grades, but the life of steels exposed to alternate stresses which exceed 40,000 lb. per square inch has been found to be comparatively short.

Brittleness of fracture, in all grades of steel, is characteristic of fatigue tests. Brittleness under repeated stresses, therefore, does not constitute evidence of defective or inferior metal. Primitive toughness is of value to meet occasional overloads, at certain stages, but its value in prolonging the life of steel under repeated alternate stresses of moderate degree, if it has an influence, is not known. There are certain zones for each grade of steel within which its endurance is practically unlimited, and above which repeated stresses soon result in rupture. A fatigue fracture may be made in any piece of steel. It is only necessary that a given load shall be repeated a sufficient number of times, when rupture will ensue.

On this subject we are dealing primarily with the ability of steel to endure strains of tension and compression alternately, each of which are substantially confined to elastic movements. When the magnitude of these alternate strains does not exceed certain limits, they may be repeated a great number of times without causing rupture. Experimental research with carefully prepared bars has shown that the mag-

nitude of such strains in steels of high resistance is limited to about one and one-third thousandths of the length of the uniformly strained part, a strain which corresponds to a stress of 40,000 lb. per square inch (2800 kg. per square centimeter) Lower strains define the limiting stresses for medium grades of steel. Accepting the results of these experimental bars, it would be undesirable to submit carbon steel to repeated alternate stresses in excess of 40,000 lb. per square inch, since this stress would be expected to eventually cause rupture.

In the laboratory, a steel of about 0.85 carbon displayed the greatest endurance. There are reasons for preferring a lower carbon in rails, and with the lowering of the carbon content a lower limit of endurance would be looked for, other considerations remaining unchanged.

It would be prudent to restrict the fiber stresses in rails to less than 40,000 lb. per square inch. It is ignoring general engineering practice and precedent to apply loads to a structure having no residual strength above the working loads. The occasional application of stresses in excess of this limit constitutes a menace to the integrity of the steel, although the working loads, in general, do not reach this maximum. If it shall appear that rails are subjected to stresses in service which approach or exceed the above-mentioned fiber stress, then their failure by fatigue will present no novel feature.

Steel rails, in the track, are subjected to bending stresses, the magnitude of which depends primarily upon the amount of the wheel loads; the number of repetitions within a given interval of time depending upon the density of traffic. The values of the static wheel loads admit of accurate determination. It is known that such loads are greatly exceeded when the trains are in motion.

Concerning the fiber stresses in the track under slowly moving locomotives, observations were made by the writer, in 1893 and 1894, the results of which were published in the reports entitled "Tests of Metals." Even under the lighter equipment then in use, it was found "that rails were often strained higher than material is supposed to be in the case of bridges and other permanent structures." It was then brought out that wheel spacing influenced the results; that the leading pilot wheel caused much higher stresses, per ton of wheel load, than other wheels of the locomotive; that tie spacing did not afford the basis for computation of the fiber stresses, the wheel load and the moment of resistance of the rail being known; and that different kinds of ballast influenced the results.

At that time a rail was selected, representing good track conditions, and measured on frozen gravel ballast. It showed a variation in measured strains in different parts of its length of over 100 per cent., referring to the effect of one of the driving wheels. The aggregate longitudinal tension from eight wheels, at one place on the rail, was nearly three times that at another place along its length, while the compressive stresses dis-

played at one place were 10 times those measured at another place. The measured strains, although taken in the track 22 years ago, should effectually dispel any illusions that rails are uniformly strained, even under static conditions of loading.

In these early tests, the fiber stresses ranged from 10,000 to 18,000 lb. per square inch (700 to 1260 kg. per square centimeter) under the heavier wheels of the locomotives. There are more recently published accounts of fiber stresses, measured under trains traveling at moderately high rates of speed, the values of which range from 25,000 to 30,000 lb. per square inch (1750 to 2100 kg. per square centimeter), with higher maxima in occasional observations, in tension the stresses being greatest in the base.

Rails in service not infrequently show appreciable local bends, acquired after they have been in the track. These bends represent permanent sets given the rails by loads which have exceeded the elastic limit of the steel. Some rails are bent downward at their ends, others are bent at regular intervals corresponding to the length of the peripheries of driving wheels of locomotives which have been over the rails, the latter permanent bends being attributable to the dynamic augment of the drivers.

There are internal strains in rails which result from conditions attending their manufacture, and those which are acquired after the rails have reached the track. During fabrication, cooling strains are acquired. The shape of a rail is such that it is peculiarly susceptible to the acquiring of strains under ordinary conditions of cooling. The stage at which these cooling strains are chiefly acquired is after the rail has left the last pass of the rail mill, and has become a finished rail.

Normally, the parts first to cool are temporarily put into a state of internal tension, which is reversed to a state of compression when the entire mass has reached atmospheric temperature. When the rail is cold, it is common to find the flanges in a state of internal compression, with the metal along the middle of the base in a state of tension. At the junction of the web with the head and the base, a state of tension is common. The peripheral metal of the head is left in compression, with the interior of the head in tension. The cooling strains are usually much greater in the base than in the head of the rail. Cooling strains in the flanges of thin-flange rails have been found as high as 18,000 lb. per square inch compression (1260 kg. per square centimeter), with metal in close proximity at the middle of the base in a state of tension. Longitudinal strains are referred to in these remarks. It is obvious that shearing strains will be set up by these contiguous internal strains of tension and compression.

Accelerated air cooling increases the magnitude of the cooling strains. The normal relations between the internal strains in the head and the base may be reversed by a sudden quenching of the rail. For the latter purpose it is not necessary to quench the rail from a temperature as

high as 1000° F. The introduction of cooling strains does not involve an approach to the temperature of the recalescent periods.

The usual initial state of compression in the flanges may be reversed to tension by cold-bending the rail, accomplished by an overload of compression on the base. Gagging is merely applying an overload, causing a permanent set in the rail, and setting up new internal strains for the purpose of initially straightening it. It disturbs the primitive state of strain left by cooling, leaving the rail in a state of critical equilibrium from which it is again easily disturbed by track conditions.

When the rail reaches the track, cold-rolling of the running surface of the head takes place under the action of the wheel pressures. The top of the head then acquires a higher state of internal compression than left by normal cooling, the strains in the head acquiring an ascendency over those of the base. Internal strains of compression next the running surface have been found in rails from the track corresponding to over 20,000 lb. per square inch stress (1400 kg. per square centimeter).

Internal strains from wheel loads are unavoidable. The hardest rails are not exempt from this action of the wheels, nor do the lower wheel loads fail to introduce internal strains. An experimental inquiry into the effects of wheel pressures showed that a wheel load of 15,000 lb. (6800 kg.) introduced strains that were higher, superficially, than those resulting from the higher wheel loads of 25,000 and 35,000 lb. (11,340 and 15,875 kg.). It was also noted that rerolling with 35,000 lb., after lower wheel pressures had been used, resulted in diminished internal strains over those due to the lower pressure. Furthermore, peining the surface with a light hand hammer introduced internal strains of greater magnitude than witnessed in either the experimental rails or those from the track.

In this connection, it is necessary to take into account the volume of the metal directly affected by the cold-rolling, and also the effect of repeated alternate stresses on the physical properties of the steel. The depth of penetration was found to be greater with the higher wheel loads than with the lighter loads or with the peining. The ratio of the volume of metal directly affected by the cold-rolling to that of the reacting metal is also a matter for consideration. The interior of the head is put into a state of tension to balance the volume of metal which is in a state of compression. Tensile stresses in the interior of the head, equivalent to the measured strains, have reached values of 8000 to 9000 lb. per square inch (560 to 630 kg. per square centimeter).

Experiments of this kind do not afford opportunity to observe the changes in physical properties which attend the application of alternate stresses, when those stresses are overloads, causing permanent sets in the steel. This feature, however, was covered in some tests made at the Watertown Arsenal in 1889 and subsequent years. It was then found

that an overstraining load in one direction impaired the elastic limit in the opposite direction; that is, an overload in tension lowered the value of the elastic limit in compression, and *vice versa*. The modulus of elasticity was temporarily lowered and a variable value displayed in overstrained steel. These are features which no doubt exert an influence on the affected zone of metal adjacent to the running surface of the head of the rail.

A rail of sufficient hardness to retain the shape of its head is well adapted to acquire and retain internal strains. Lateral flow of the metal necessarily tends to relieve longitudinal strains. The presence of a fin at the side of the head is evidence of lateral flow having taken place, and this flow should be attended with some relief to the longitudinal strains. The relation between direct and crosswise strains expressed by Poisson's ratio, having reference to the effect of external loads, does not prevail in cases of internal strains set up during cooling or resulting from cold-rolling. Compression of the metal in one direction is not attended with negative strains at right angles thereto, in cases of internal strains. A state of internal compression may exist in both lengthwise and crosswise directions.

Of the two components which tend to cause fatigue fractures in steel rails, it will be borne in mind that one of them, namely, the state of internal compression at the running surface of the head, and the concomitant state of longitudinal tension in the interior of the head, will be introduced by the lighter wheel loads as well as by the heavier ones. These internal strains act upon the entire length of the rail. The direct bending stresses, the other component, will depend upon the wheel loads. If not strictly proportional to the wheel loads, still the bending stresses will be greater as the wheel loads are increased. Again stated, the two components which tend to cause fatigue fractures are the direct bending stresses from the wheel loads, and the internal strains introduced by the cold-rolling of the wheels on the head of the rail. The latter are responsible for a disturbance in the physical properties in addition to the introduction of internal strains.

It is of importance to acquire data upon the relative effect of these two components in an endeavor to ameliorate conditions and prolong the life of the rail. Interest centers upon that influence which causes the maximum longitudinal tension, since a fatigue fracture is one of tension. The early display of transverse fissures in rails of 125-lb. weight (62 kg. per meter) leads to the inference that the cold-rolling component is of grave importance, which may be greater relatively than the direct bending stresses. There have been examples of rails sustaining traffic after the head had been practically separated by the development of a transverse fissure. The web and base of the rail, for the time, resisted the bending stresses, illustrating the ability of the steel to endure very high fiber stresses, for a limited number of repetitions.

Primarily, it is desirable to ascertain, by means of direct experimental inquiry, the ability of steel rails to endure repeated alternate stresses, acquiring this information upon the rail in the condition in which it comes from the rail mill, and before any cold-rolling by wheels in the track has taken place. The section of the rail is such that opposing cooling strains are necessarily present in the finished rail. There is no assurance that a rail in full cross-section will display the endurance under repeated alternate stresses which carefully prepared bars of the laboratory have shown. Tests of this general nature on rails of full cross-section have yielded results which emphasize the need of this information.

Next, it is important to ascertain the limit of endurance of rails which have been in the track and have acquired a state of internal strain. There is no apparent reason why this barrenness of fundamental data upon steel rails should longer continue. In the use of materials of construction, two features should be known: The ability of the material to endure stresses, in the form in which it is used; and knowledge of the magnitude of the stresses which are to be endured. The steel-rail problem presents an example in which neither of these basic considerations has been definitely known. This dearth of relevant information is mentioned since its absence has led to many unsupported conjectures as to the causes of rail failures.

Referring to the location in the rails in which transverse fissures are prevalent: they occur in the head, and not in the web or the base. They commonly make their appearance on the gage side of the head, or central, over the web. In a lot of 663 transverse fissures, 535 were located on the gage side of the head, 128 over the web, and none on the outside of the head. The preponderance of transverse fissures on the gage side of the head is significant, of course. Conditions of loading commonly reach a maximum on the gage side of the head.

It has been found that transverse fissures, made by progressive gagging of new rails, can be located at will on the right or the left side of the head, or central, according to the manner of applying the load. It is only necessary to apply the load to the side of the head on which it is chosen to locate the fissure, to accomplish that result.

Twenty-four transverse fissures were experimentally made in new rails by progressive gagging. By progressive gagging, is meant that the rail was gagged at close intervals along its entire length, first on the head and then on the base, repeating the process until rupture ensued. The number of blows required to be struck at any one place ranged from 800 to 4000. The total number per rail length of 33 ft. was from 50,000 to 250,000 blows.

The bending stresses given the rails in these tests were in excess of those experienced in the track, a circumstance which led to their rupture before the usual burnishing effect was accomplished which the opposite

faces have upon each other in producing a silvery luster; rupture also being completed before the nuclei of the fissures had extended and covered a large part of the cross-section of the head.

Efforts to produce transverse fissures by repeated gagging in one plane only were unsuccessful. Such rails displayed the common type of fatigue fracture; that is, the fractures had exterior origins in each case. Progressive gagging has an effect akin to the cold-rolling of the wheels

In rails from the track, transverse fissures have been found, along the same element of the head, in different stages of development. This manifestation corresponds to the shattering effect of repeated alternate stresses witnessed in laboratory test bars which have been exposed to a uniform bending stress over a part of their length.

Critical examinations have shown transverse fissures to have their origins in metal microscopically sound and normal in structure. Diligent research has failed to reveal micro-defects to which the origins of the fractures could be attributed. The composition of the steel at and in the vicinity of the fissures has shown no chemical reason why the fissures should have their origins at the places where they were found.

There has been no example of a transverse fissure in a rail which has not been in service, excepting experimental fissures. Rolling conditions would preclude the presence of a transverse fissure in an advanced stage of development in a new rail. The reduction of the metal in the rolls orients slag and other inclusions, arranging them in a direction parallel to the length of the rail, and not crosswise.

It is well known that slag inclusions are present in some degree in most steels. Slag filaments are drawn out and may be detected in the cross-section of the rail. In longitudinal sections they are at times prominently shown; but such longitudinal seaminess, when it exists, has no connection with the formation of transverse fissures. Composite sketches showing longitudinal seaminess in parts of the head remote from the places occupied by the nuclei of transverse fissures necessarily have no bearing upon the question of the development of transverse fissures.

It not infrequently happens, however, that longitudinal seams of another kind are developed in conjunction with transverse fissures, wheel pressures being the common cause for each. Shearing strains are set up in the head of the rail by the wheel pressures. When an acicular streak is reached, which may represent a slag inclusion, opportunity is afforded for the starting of a longitudinal seam. Shearing fractures which develop in vertical planes are designated as split heads. In other cases, the separation takes place in horizontal or oblique planes, and these latter may be developed independently or associated with transverse fissures. Each of these types of rupture was represented in the rail which failed on the Lehigh Valley Railroad.

The effect of gagging has been referred to in respect to introducing

internal strains or in modifying those which preëxisted. Rail sections were subjected to normal and excessive gagging, also to the more severe ordeal of being struck with the tup of a drop testing machine. Microscopic examination directed to ascertain the effect, if perceptible, did not show any appreciable distortion of structure either at the places directly acted upon, or in other parts of the cross-sections of the rails. Macroscopically, the mill scale on the surface of the rail was disturbed in the vicinity of the place gagged or struck with the tup of the drop-testing machine. Surface indications comprised all that was visible resulting from this treatment.

Frequent use has been made of the strain gage in the examination of steel rails. The strain gage offers an opportunity to measure changes in dimensions not readily accomplished by other means. On a gaged length of 10 in. (25.4 cm.), it is not difficult to detect a change in length of one part in fifty thousand, and not infrequently a change of one part in one hundred thousand may be reliably shown. Under the microscope forms appear, but they are of such a character that a distortion equivalent to a stress of say 40,000 lb. per square inch (2800 kg. per square centimeter), the assumed limiting stress for steel to retain its integrity unimpaired, would escape notice, whereas a distortion of one-hundredth of this amount would be detectable with the strain gage.

The relations between the forms shown by the microscope, or the micro-constituents as they have been designated, and the ability of the steel to endure long-continued alternate stresses have not been shown. In the use of materials subjected to strains, strains or their equivalent stresses represent the fundamental features upon which it is essential to acquire data. If matters of segregation of the chemical constituents or their arrangement as indicated by the microstructure of the steel exert a controlling or collateral influence on the ability of the rail to endure service stresses, as it would seem that some relation must exist, then it is highly desirable to have this relation established; but this does not appear to have yet been done, notwithstanding the considerable use currently made with the microscope. The relation between segregation, within the range of composition covered by ordinary rail steel, and the endurance of the metal under service conditions, has not been established. It is necessary to take into account these features in discussing the causes of the fracture of steel rails, in order to properly differentiate between those which are known to exert a definite influence and those which are conjectural, in the light of present knowledge.

For the purpose of illustrating the relations between the fiber stresses at the running surface of the head of the rail and along an interior element, in the region in which transverse fissures have their origin, some hypothetical cases of loading will be presented. The values of the internal strains used are those which have been found in the measured strains of

rails that have developed transverse fissures in service. An initial state of compression at the running surface of the head of 15,300 lb. per square inch (1265 kg. per square centimeter) was found by measurement, and a state of tension of 5245 lb. per square inch (524 kg. per square centimeter) along an interior element. Each of these values is frequently exceeded in rails from the track; therefore the computed results will be taken as exhibiting conservative relations. In tabular form these computed results are as shown in Table 1.

TABLE 1.—*Hypothetical Cases of Loading a Rail in which Internal Strains Preëxisted*

| Rail under Assumed Bending Stress of Tension in Head, Pounds per Square Inch | Longitudinal Fiber Stress, Pounds per Square Inch | |
|---|---|---|
| | Running Surface of Head | Interior Element in Region of Transverse Fissures |
| No wheel load | 15,300 compression | 5,245 tension |
| 15,300 fiber stress | Zero stress | 17,485 tension |
| 30,600 fiber stress | 15,300 tension | 29,725 tension |
| 40,000 fiber stress | 24,700 tension | 37,245 tension |

It is not held that the above relations will strictly be realized, but it is believed that substantially such relations may exist between the metal at the top of the head and along an interior element, and that the interior elements will retain the state of tensile ascendency under the usual range of bending stresses which are received by the rail in the track. Under bending stresses, which are not unusual, the computed results call for a tensile stress in the interior of the head in excess of that at the running surface. The development of an interior fracture by repeated stresses under these conditions appears to represent normal behavior.

Attention has so generally centered upon the results of the drop test, the bending test, or the tensile test, in respect to the display of primitive ductility, that the phases through which rupture is reached by repeated stresses have not received adequate consideration. Specifications governing the acceptance of the metal rigorously demand that a certain elongation shall be displayed. The elongation called for may be only a few hundredths of an inch per inch of length of sample, but such display is made the decisive measure for the acceptance of the steel. It would not be incompatible with experimental results for the steel which was deficient one-hundredth of an inch in its primitive display of elongation to show under repeated stresses an aggregate elastic extension of several miles, so radically different may be its primitive behavior and its behavior under long-continued alternate stresses.

In like manner, chemical requirements are frequently made the object

of exact fulfillment. Without advancing other reasons, the cause of the failure of materials under service conditions has been attributed to some slender deficiency in fulfilling the specifications on which the material was supposed to be made. It is not always clear that the most suitable steel for the purpose is asked for in specifications. Attributing the cause of failure of a rail to the use of one deoxidizer or another, or to an excess of some non-metallic constituent above prescribed limits, can hardly carry conviction in the absence of information which connects them as cause and effect, if they stand to each other in that relation.

It is desired to emphasize the fact that the durability of a rail consists of its ability to retain its integrity under repeated deformations. A direct test consists of subjecting the rail to repeated deformations, conducted upon the shape and dimensions in which it is used. The results of other tests may be accepted as indexical of the properties desired, provided the relations between the two are established. In general, however, direct methods are preferred to indirect ones.

In order to show the diversity of views and opinions which have been expressed upon the causes of transverse fissures, the following extract is made from the report of the Committee on Rails and Equipment, of the National Association of Railway Commissioners, at its twenty-seventh annual convention, San Francisco, Cal., October, 1915.

"Summarizing the causes ascribed as responsible for the formation of transverse fissures, * * * we have the following:

1. Unsoundness of the ingot, blow holes and piping.
2. Segregation in the ingot.
3. Rolling green ingots.
4. Rolling strains of the rail mill.
5. Direct rolling from the ingot to the rail. (Conversely, avoiding transverse fissures by using reheated blooms.)
6. Angular friction resulting from the coning of the wheels.
7. Greater prevalence in open-hearth than Bessemer steel.
8. Incomplete transformations in the steel, interior metal of the head being left in the austenitic form.
9. Conditions at the hot bed.
10. Gagging the rail, in straightening.
11. The position of the rail in the ingot.
12. Rails rolled in the winter months most likely to develop transverse fissures.
13. The direction they pass through the rail mill to be considered with reference to the direction of traffic.
14. Some latent tendency introduced in the steel, in the process of manufacture, which track conditions render active.
15. Inherent weak portions, such as porosity, sponginess and slag inclusions.
16. That certain heats are more susceptible than others.
17. That certain year's rollings were subject to suspicion.
18. That transverse fissures develop most rapidly when, under atmospheric conditions, the rail is expanding and contracting.
19. With summer temperatures, when the rails are fully expanded, increased thermal stress augments the internal strains of fabrication.

20. Another says, concerning summer months, that the strains are greatly relieved by reason of having reached their full expansion.

21. That rails under summer temperatures can better adapt themselves to reversals of stresses.

22. That there is a predominance of transverse fissures in rails of the heavier sections.

23. That the carbon should be limited in rails of 100 lb. and under.

24. That traffic conditions do not influence the development of transverse fissures.

25. That traffic conditions do influence their development."

The statement that a transverse fissure is a modified type of fatigue fracture is based upon the identity of the manifestations which attend and the conditions which lead to rupture in each case, and furthermore upon the absence of data leading to other explanation.

In conclusion, a quotation will be presented from a recent report issued by the Interstate Commerce Commission, Division of Safety, embodying remarks by the writer on the investigation of a rail which failed in the track, displaying a number of transverse fissures:

"Neither chemical analyses nor microscopic examinations have shown a definite cause for the development of transverse fissures. They have occurred in rails of different weights; in those of different ingot positions; in the product of each of the two great methods of steel making, Bessemer and open-hearth; in direct-rolled rails and in those from reheated blooms; over ties and between them; they have made their appearance in the different seasons of the year, and in rails rolled in the different seasons; they occur where, according to chance, the rails may have been gagged and from the number and proximity of the fissures, where they probably were not gagged, they have displayed themselves singly, and independent of other types of rupture, and also associated with shearing fractures or seams which have developed in planes at right angles to the fissures; they are found in the head and not the base of the rail; they appear in rails which show very little wear; they are not confined to any one rail mill, nor to northern or southern mills, nor to any one year's rolling, but have been displayed by rails rolled at periods of time separated by not less than two decades. The latter fact should remove doubts concerning any peculiar conditions having found their way into modern mill practice influencing this type of failure.

"On the other hand, traffic has increased, wheel loads have increased, and speeds have increased; and transverse fissures are in general found where these conditions are well advanced. The girder strength of the rails has been increased, while the impinging pressures of the wheels still remain without amelioration. It is a disquieting matter that rails of the heavier sections recently laid have displayed transverse fissures. Mere increase in weight of section has not brought with it immunity from failure in this manner. It gives emphasis to the need of acquiring further data upon the features which have been discussed in this report.

"The conclusion seems well founded that transverse fissures are fatigue fractures, and that they develop in rails which are structurally and chemically free from any known defect. It is a modified type of fatigue fracture in which there is a compressive component in the rail next the running surface of the head. The presence of this compressive component accounts for the interior origin of the transverse fissure. It constitutes the difference which the introduction of the term 'transverse fissure' was intended to emphasize over the common type of fatigue fracture in which this component is absent and which in consequence thereof has an exterior origin."

## Discussion

H. D. Hibbard, Plainfield, N. J. (written discussion*).—This is an extremely valuable and timely paper and if it does not go to the root of the matter, it deals with many of the elements and factors involved. It might be taken as the engineer's solution of the case, starting with the assumption that steel is steel, while the tendency for a rail to have transverse fissures may, and probably does, depend, in part at least, on conditions and things met before it reaches the track.

Admitting with the author that "brittleness of fracture in all grades of steel is characteristic of fatigue tests," we may yet look farther back to find out, if possible, why some rails and some steels are more prone to have transverse fissures than others, for it has been found that rails from certain heats of steel developed these fissures under the same service which did not fissure rails of other heats. That demonstrates that some steels resist the fissuring conditions better than others, or to put it otherwise, that they are better.

It seems to the writer that a quickly developed transverse fissure in a rail results from a combination of causes which begin with the treatment of the molten metal in the steel furnace and end with the fissure of the rail in the track. Briefly, these are poor steel, massive head, internal stresses in the head, and, given such conditions, to the wheel action in bending the rail and stretching the top surface metal of the head by cold rolling, as the author has described, the tendency being aggravated by gagging, which may at least determine the location of the fissures, or some of them.

Thus the service stresses, instead of being the fundamental cause, may be, as it were, the last straw, and if the steel were good and the internal stresses in the head were absent, might not be great enough to fissure the rail.

The quality of the steel is determined largely by the way it is finished in the furnace, a well worked, readily killed bath, not easily described in words, being wanted, which demands more time than furnacemen, who are paid for the tonnage produced, are sometimes willing to allow. Then sufficient time must be allowed after the final additions for the oxides and silicates (sonims) resulting from the consequent reactions at least to agglomerate into drops, and preferably enough time for the drops to rise to the top.

The heating and rolling which, of course, must be properly done are now rarely responsible for poor steel and presumably for transverse fissures. Nevertheless, it is quite conceivable that in rapid heating and rolling ingotism is not wholly obliterated and that carbides, sulphide or phosphide any or all remaining undiffused between the grains weakens

* Received Jan. 15, 1918.

the metal. The additional time for this afforded when the blooms are reheated before going to the finishing mill may cause the diffusion of more of these ingredients, thereby strengthening the steel, and so justify in a measure the opinion of those who think such reheating prevents transverse fissures.

The greater prevalence of fissures in open-hearth steel may be fairly laid to the more massive heads, which became common at about the same time that the use of open-hearth steel for rails became general, rather than to the process. The interior of the head of a rail, which cools last, is in state of tension, as Howard and others have found; this is true of the interior of every piece of steel cooled in the open, and the more massive the piece the greater the tension. When the stresses from wheel action, added to that initially in the central part of the head, exceeds the strength of the metal, fissuring begins which grows under continued service. The better the steel, the better it will resist.

If the explanation of the genesis of transverse fissures here given be correct, the cure of the trouble, or at least its decrease to the minimum, is fairly obvious:

First, have the steel well made, so that it will pass a more severe drop test than is now prescribed; this will cause the rejection of heats poorly made and therefore liable to quickly have transverse fissures.

Second, heat properly and finish the rolling of the rail when wholly at a temperature near, but above, the recalescence point.

Third, retard the cooling of the rail after rolling so that it shall cool at a substantially uniform rate throughout; this will prevent internal stresses in the head and elsewhere. Rails to be so cooled would be hot-straightened and would not require to be cambered after rolling, as is necessary when they are to cool in the open air according to present practice. Then cold-straightening would be diminished, if not avoided.

Of course, rails thus dealt with could not be heat-treated by any process involving accelerated cooling unless it were followed by annealing. Rails so made could safely contain higher carbon than is now customary, and for that reason would better resist cold-flowing and stretching of the surface. Heavier sections could also be employed, which would better endure the wheel action, which contributes toward the development of transverse fissures.

C. W. Gennet, Jr.,* Chicago, Ill. (written discussion†).—Since the Lehigh Valley accident, transverse fissures have become a source of constant anxiety to railroad officials, because such defects, only infrequently detected by the trackmen, may first appear under trains entirely without warning. The number of actual accidents resulting

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* Engineer with Robert W. Hunt & Co.

† Received Feb. 9, 1918.

from rails containing fissures is no doubt a small part of the whole, but the large losses that have been directly attributable to fissures, coupled with the continual possibility of repetition, creates an alarming situation which demands a full investigation of the cause of fissures, apart from other types of rail failures. Mr. Howard's position gives him a field for action and an opportunity for laboratory examination and research that are unsurpassed, and the results of his investigations must be received with the utmost attention and respect. His admirable paper, summarizing his previously published studies and analysis of the transverse fissure problem, constitutes a convincing treatment of the subject chiefly from the standpoint of stress and strain to which rails are subjected. The intricacies of the problem are so great that seemingly any theory advanced for the cause of fissures may be attacked from some angle; and although Mr. Howard's deductions are logical in the direction followed, it is apparently necessary for considerable work to be done along other lines before his theories can be fully accepted.

It is generally conceded that fissures, whether fundamentally due to fatigue or something else, are of a progressive character, their ultimate size being the result of growth from an originating nucleus or point of rupture. Abundant proof of this lies in the variable sizes of the fissures found, and there can be no doubt that their development or growth is the result of the strains to which rails are subjected in the track. Obviously, therefore, the most important factor in investigating the cause for fissures lies in determining the conditions that exist at the point of original rupture, that is where the separation of the metal at the nucleus occurs. The composition of the steel chemically and structurally in this small area thus becomes of vital interest, and it is not sufficient to place too much credence in the manifestation of the good conditions that may be found elsewhere in the sample under investigation.

The inferior conditions that may be present not only at the original point of rupture, but at other places, are manifold and minute, and examination of them requires careful and skilful work by trained metallurgists and microscopic observers. Absence of results of critical examinations of various specimens in many of Mr. Howard's Government reports, including his present paper, is surprising and disappointing and does not carry conviction to the statement that "Critical examinations have shown transverse fissures to have their origin in metal microscopically sound and normal in structure." In fact, the results of some chemical analyses of steel containing transverse fissures that have been reported are such as to direct prompt suspicion to the quality and consequent structure of the steel; and that "Neither chemical analyses nor microscopic examinations have shown a definite cause for the development of transverse fissures" is due perhaps to insufficient chemical and microscopic work to establish such connection. The question is whether Mr. Howard's

studies have been directed along certain metallurgical lines far enough to justify, beyond argument, his statements which plainly relieve the chemical and physical condition of the steel from any responsibility regarding the origin of fissures.

Several reports have been issued by the Interstate Commerce Commission giving the results of Mr. Howard's work in connection with accidents caused by transverse fissures. The following table gives the average results of the various chemical analyses made on rails containing fissures mentioned in those reports.

| Rail | C | Mn | P | S | Si | Ni | Cr |
|---|---|---|---|---|---|---|---|
| A | 0.843 | 1 20 | 0 042 | 0.032 | | | |
| B | 0 860 | 0.67 | 0 050 | 0.033 | 0 016 | | |
| C | 0.690 | 0.79 | 0 095 | 0.025 | 0 005 | | |
| D | 0.830 | 0.79 | 0.062 | 0 041 | 0.165 | 0 066* | 0.05 |
| E | 0.840 | 0.79 | 0.059 | 0.040 | 0.152 | 0 280 | 0 02 |
| F | 0.801 | 0 91 | 0.015 | 0.041 | 0.125 | | |
| G | 0.710 | 0 77 | 0.121 | 0.030 | 0.003 | | |

None of the above rails, possibly excepting F, is acceptable under the requirements governing the chemical composition laid down by recognized specifications for rail steel. The "slender deficiency" by which most of these steels fail to fulfill common rail specifications is not apparent, and although "it is not always clear that the most suitable steel for the purpose is asked for in specifications" (probably meaning rail specifications) still the burden of thought is largely against the advisability of using steel of the hardness and brittleness indicated by the above analyses. That such steels have been accepted and put into track unknowingly, of course, and in spite of the specifications, is a criticism of the methods of procedure and by no means a defense for that steel when fracture later occurs.

Aside from being chemically defective, in a general sense, for use in rails, the above steels offer other significant features for consideration. High carbon coupled with high phosphorus makes a hard non-ductile steel to start with, but the effect that large amounts of these elements, singly or together, may have in small local areas deserves study. Further, the low silicon in some of the steels above mentioned indicates the liberal use of aluminum at the time the steel was cast into ingots. Large amounts of aluminum added to steel, especially in the molds, has often been regarded as doubtful practice, while the resulting presence of alumina in soft steel has been identified and the good qualities of the steel questioned. Recently alumina has been microscopically found in rail steel, and incidentally in rails containing transverse fissures.

* Probably incorrect.

Basic open-hearth steel, such as used for rails, is easily subject to the many vicissitudes of heat treatment; in fact, rails are virtually heat treated by the action they undergo when cooling on the hot beds. The influence there of cold winds and contact with the cold skid rails may be marked. Some effects of unusual heat treatment may be, and doubtless are, confined to short lengths of rail, and it is almost inconceivable that austenite or cementite does not exist locally in many cases. The presence, therefore, of distinctly non-ductile localities or regions is plainly predicated. All rails containing these hard spots caused by either chemical or physical conditions must be subjected, of course, to the damaging blows of the straightening presses long before they ever reach the track and apparently opportunity is thus offered for interior injury to occur.

Slag inclusions are held responsible by Mr. Howard for the chief type of rail failures, those known as split heads, his theory being that the inclusions are elongated in the process of rolling and become streaks in the metal which are unable later to resist the shearing action of the traffic. But hard spots of a non-ductile character, originating perhaps from chemical causes or possibly minute globules of a distinctly foreign nature, or from the physical effects produced by the methods by which rails are cooled, are not regarded as menacing the integrity of the metal, merely because the presence of such conditions has not been even perfunctorily identified in the rails examined, notwithstanding the indications, and the definite proof of their existence in certain other cases.

Thus it seems inconsistent now to attribute the cause of fissures to simple fatigue fracture for which wheel loads are mostly responsible, and the suggestion is advanced that as microscopic work proceeds the effects of sonims, non-ductile spots, and the use of steel chemically unsuited for the purpose, may be more fully established; in the meantime, the situation emphatically warrants a suspension of judgment until painstaking studies can be made.

It seems regrettable that with a subject so important as that of transverse fissures no particular effort has been made to accumulate systematically the important historical data pertaining to the various cases. Obviously this is a work for the Railroad Engineering Association, whose Engineer of Tests should have a complete index of the different cases, with such information on each as would make it unnecessary to deal with generalities, as is now so often the case. Such cataloging of the fissures that have occurred should show, as accurately as possible, among other things:

1. Name of manufacturer and railroad.
2. Dates of rolling, and occurrence of fissure.
3. Heat number and chemical analyses (complete).
4. Location of rail in the ingot (and, if possible, the ingot in the heat).

5. Location of the fissure in the rail with respect to the branded side of the rail.

6. Location of the fissure in the rail with respect to the track-gage side of the rail.

7. Was rail on high or low side of a curve or on straight track?

With the results of such a tabulation covering a large number of cases, the study of certain matters concerning fissures could be much better approached than now, when comparatively few and rather isolated cases must usually be considered.

John D. Isaacs,* New York, N. Y. (written discussion†).—In brief, Mr. Howard bases his conclusion upon the following facts:

1. The law of fatigue of metal was established 50 years ago. Repeated reversed stresses above 30,000 lb. per square inch (2100 kg. per square centimeter) cause failure of iron; and above 40,000 lb. cause failure of steel.

2. Steel rails in track are subjected to repeated alternate bending stresses, the magnitude of which depends upon magnitude of wheel loads.

3. Cooling stresses exist in rails from the time of manufacture, these are compression on surface and tension in interior.

4. Cold-rolling in service increases the surface compression.

5. Eighty per cent. of transverse fissures have occurred on gage side, 20 per cent. over web and none on outside.

6. Fissures have been located at will on right or left side or central in the head by progressive gagging.

7. In the metal surrounding transverse fissures, neither microscope nor chemical analysis show anything unusual.

8. No other theory as to the cause of transverse fissures has been proved.

From these facts Mr. Howard draws the following conclusion: Transverse fissures are fatigue fractures caused by excessively great repeated reversed stresses. He recommends that fiber stresses in rails be restricted to less than 40,000 lb. per square inch.

The following facts appear to controvert this conclusion:

1. Transverse fissures have occurred in rails which showed very little wear.

2. Mere increase in weight of rails has not brought immunity from transverse fissures and rails of the heavier sections have displayed transverse fissures after short service.

3. Transverse fissures are very rare (Southern Pacific Co.'s reports show one failure in 114,000 rails per year). If fatigue were the cause, all of the old rails in heavy service should be breaking up from this cause.

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* Cons. Eng., Southern Pacific Co.

† Received Feb. 15, 1918.

4. Transverse fissures are more prevalent in the products of some mills than in those of others.

5. The range of repeated stresses is greatest at the surface of the rail, as shown by Mr. Howard's table, which may be extended as shown in the accompanying Table 1.

TABLE 1.[3]—*Hypothetical Cases of Loading a Rail in which Internal Strains Preëxisted*

| Rail under Assumed Bending Stresses in Head, Pounds per Square Inch | Longitudinal Fiber Stress, Pounds per Square Inch | |
|---|---|---|
| | Running Surface of Head | Interior Element in Region of Transverse Fissures |
| 40,000 lb. compression | 55,300 lb. compression | 26,755 lb. compression |
| 30,600 lb. compression | 45,900 lb. compression | 19,235 lb. compression |
| 15,300 lb. compression | 30,600 lb. compression | 6,995 lb. compression |
| No wheel load | 15,300 lb. compression | 5,245 lb. tension |
| 15,300 lb. tension | Zero stress | 17,485 lb. tension |
| 30,600 lb. tension | 15,300 lb. tension | 29,725 lb. tension |
| 40,000 lb. tension | 24,700 lb. tension | 37,245 lb. tension |
| Range of repeated stresses | 80,000 lb. or 125 per cent. | 64,000 lb. or 100 per cent. |

It is seen that the range of stresses at the region of transverse fissures is 64,000 lb. and 25 per cent. more than this, or 80,000 lb. at the surface of the head. The effect of fatigue of metal should be greatest where the range of stress is greatest and if the rail failed from this cause, the break should start at the surface. This disposes of the principal argument upon which Mr. Howard's theory is based.

6. Mr. Howard could not locate transverse fissures "at will" lengthwise of the rail by progressive gagging nor could he produce transverse fissures in any specific plane, indicating that some inherent defect starts the fissure. Transverse fissures were developed by him by groping for the weak places—progressive gagging.

7. Service is required to develop any sort of a defect into a failure.

8. The relation that exists between wheel loads and track stresses has not been even approximately determined. Limitation to a definite intensity is therefore impossible at present.

*Conclusions.*—Mr. Howard's theory that tranverse fissures are fatigue fractures caused by excessive and repeated reversed stresses has not been proved and cannot be true in the face of the above facts. A limit cannot be put upon the fiber stresses in rails in track until the relation of wheel loads and track stresses has been established beyond a doubt. Until this relation is known, the present practice, that of laying rails which experience shows will satisfy the traffic conditions, must be

[3] Compare Table 1 of Mr. Howard's paper.

continued and reliance must still be placed upon failed-rail statistics, to show where caution is necessary and where reasonable safety is not provided.

Robert Trimble,* Pittsburgh, Pa. (written discussion†).—As I understand Mr. Howard's paper, the whole trend of it is to the effect that transverse fissures are entirely due to overstress caused by the wheel loads imposed upon rails in service. The following statements appear in his paper: (*a*) "A fatigue fracture may be made in any piece of steel. It is only necessary that a given load shall be repeated a certain number of times when rupture will ensue." (*b*) "If it shall appear that rails are subjected to stresses in service which approach or exceed the above-mentioned fiber stresses (40,000 lb. per square inch) then their failure by fatigue will present no novel feature." (*c*) "The conclusion seems well founded that transverse fissures are fatigue fractures, and that they develop in rails which are structurally free from any known defect." I agree with statements (*a*) and (*b*), but I do not admit that the proper conclusion has been deduced. It is true, no doubt, that if rails had not been subjected to service in the track they would not have developed transverse fissures in the form that we now find them.

I submit herewith Table 1 and a corresponding graphic chart containing information that I have accumulated regarding some rails manufactured by two steel companies, and laid in the main tracks of a railroad system under as nearly uniform conditions as it is possible to find, with reference to alignment, grades, roadbed and traffic conditions.

All of the rails referred to in the statement and chart were manufactured by the open-hearth process under the same specifications; that is, for any one year both mills used the same specifications. The rails are all 100 lb. per yard, and of a uniform section throughout. In regard to service, if there was any difference at all it was in favor of the rails rolled at mill A.

Mill A furnished, during the period, 83,100 tons, of which 108 rails have been removed because of the discovery of transverse fissures.

Mill B furnished, during the same period, 101,300 tons, of which 12 rails were removed because of the discovery of transverse fissures.

A study of the table and chart shows:

1. A relatively small number of failures of mill B rails compared with mill A rails; mill A has six times as many failures per 1000 rails as mill B.

2. Mill A rails have disclosed failures in 13 out of 33 rollings; and mill B rails have disclosed failures in 5 out of 30 rollings.

3. Mill A rails furnished prior to February, 1913, show that 13 rollings out of 18 have disclosed transverse fissures; while mill B rails furnished

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* Chief Engineer, Maintenance of Way Department, Pennsylvania Lines.

† Received Feb. 16, 1918.

TABLE 1.—*Transverse Fissures in Rails Rolled by Mills A and B*

| Time of Rolling | Steel-works A | | | | Steel-works B | | | |
|---|---|---|---|---|---|---|---|---|
| | Tons Rolled | No. of Fissures | Fissures per 1000 Rails | Avg Life Fissured Rails * | Tons Rolled | No of Fissures | Fissures per 1000 Rails | Avg Life Fissured Rails * |
| 1909, Feb. | .... | .... | ..... | .... | 3,100 | | | |
| Mar. | 2,600 | 1 | 0.2 | 6.7 | 300 | | | |
| Year 1909 | 2,600 | 1 | 0.2 | 6.7 | 3,400 | | | |
| 1910, Jan. | 1,500 | 3 | 1.0 | 6.1 | | | | |
| Feb. | 4,600 | 1 | 0.1 | 6.6 | 2,900 | | | |
| Mar. | 2,500 | 27 | 5.3 | 5.8 | 1,000 | | | |
| Apr. | 2,800 | 9 | 1.6 | 5.9 | | | | |
| May | 1,700 | 33 | 9.5 | 5.9 | 1,500 | | | |
| Year 1910 | 13,100 | 73 | 2.7 | 5.9 | 5,400 | | | |
| 1911, Feb. | 1,900 | 1 | 0.3 | 5.8 | 5,400 | 7 | 0.6 | 4.5 |
| Mar. | 3,000 | 11 | 1.8 | 4.9 | 700 | 2 | 1.4 | 4.8 |
| Apr. | 900 | 2 | 1.1 | 6.0 | | | | |
| Year 1911 | 5,800 | 14 | 1.2 | 5.1 | 6,100 | 9 | 0.7 | 4.6 |
| 1912, Mar. | 800 | | | | | | | |
| Apr. | 1,800 | ..... | ..... | ..... | 2,300 | | | |
| May | 5,600 | 1 | 0.1 | 2.5 | | | | |
| June | 1,700 | ..... | ..... | ..... | 2,100 | | | |
| July | ..... | ..... | ..... | ..... | 900 | | | |
| Aug. | 1,100 | ..... | ..... | ..... | 1,900 | 1 | 0.3 | 3.1 |
| Sept. | 300 | ..... | ..... | ..... | 2,300 | | | |
| Oct. | ..... | ..... | ..... | ..... | 1,300 | 1 | 0.4 | 5.7 |
| Nov. | 5,000 | 6 | 0.6 | 4.2 | | | | |
| Dec. | 4,200 | 11 | 1.3 | 3.7 | | | | |
| Year 1912 | 20,500 | 18 | 0.4 | 3.8 | 10,800 | 2 | 0.1 | 4.4 |
| Years 1909 to 1912 | 42,000 | 106 | 1.2 | 5.4 | 25,700 | 11 | 0.2 | 4.6 |
| 1913, Jan. | 3,000 | 2 | 0.3 | 2.5 | | | | |
| Feb. | 100 | ..... | ..... | ..... | 2,500 | | | |
| May | ..... | ..... | ..... | ..... | 2,100 | | | |
| July | ..... | ..... | ..... | ..... | 400 | 1 | 1.2 | 1.2 |
| Oct. | 100 | ..... | ..... | ..... | 200 | | | |
| Dec. | ..... | ..... | ..... | ..... | 1,500 | | | |
| 1914, Aug. | 1,800 | ..... | ..... | ..... | 5,200 | | | |
| Sept. | 10,100 | ..... | ..... | ..... | 7,600 | | | |
| Oct. | 500 | | | | | | | |

* At time of removal from track.

TABLE 1.—*Transverse Fissures in Rails Rolled by Mills A and B*—(*Continued*)

| Time of Rolling | Steel-works A | | | | Steel-works B | | | |
|---|---|---|---|---|---|---|---|---|
| | Tons Rolled | No. of Fissures | Fissures per 1000 Rails | Avg Life Fissured Rails | Tons Rolled | No of Fissures | Fissures per 1000 Rails | Avg. Life Fissured Rails |
| 1915, July | 2,400 | | | | | | | |
| Aug. | 1,300 | ..... | ..... | ..... | 4,700 | | | |
| Sept. | ..... | ..... | ..... | ..... | 6,500 | | | |
| Oct. | 1,900 | ..... | ..... | ..... | 10,900 | | | |
| Nov. | 4,100 | | | | | | | |
| Dec. | 2,900 | ..... | ..... | ..... | 6,700 | | | |
| 1916, Jan. | 4,000 | ..... | .... | ..... | 4,300 | | | |
| Mar. | 1,000 | ..... | ..... | ..... | 3,300 | | | |
| May. | ..... | ..... | ..... | ..... | 5,900 | | | |
| July | ..... | ..... | ..... | ..... | 1,500 | | | |
| Sept. | ..... | ..... | ..... | ..... | 2,000 | | | |
| 1917, Jan. | 3,300 | ..... | ..... | ..... | 10,300 | | | |
| Feb. | 3,300 | | | | | | | |
| Apr. | 1,300 | | | | | | | |
| Totals...... | 83,100 | 108 | 0.6 | ..... | 101,300 | 12 | 0.1 | |

during the same period show that 4 rollings out of 13 have disclosed transverse fissures.

4. Mill A shows great variation in the results from its rollings. One rolling, 1700 tons in May, 1910, showing 33 fissured rails; a rolling of 2600 tons in March, 1909, which had been in service 14 months longer, shows one fissured rail; and in another case of 4600 tons rolled in February, 1910, there is but one fissured rail.

5. The rails from mill A show a very much more frequent occurrence of transverse fissures than the rails from mill B, with a correspondingly wider range of variation in regard to the failures developed in the various rollings.

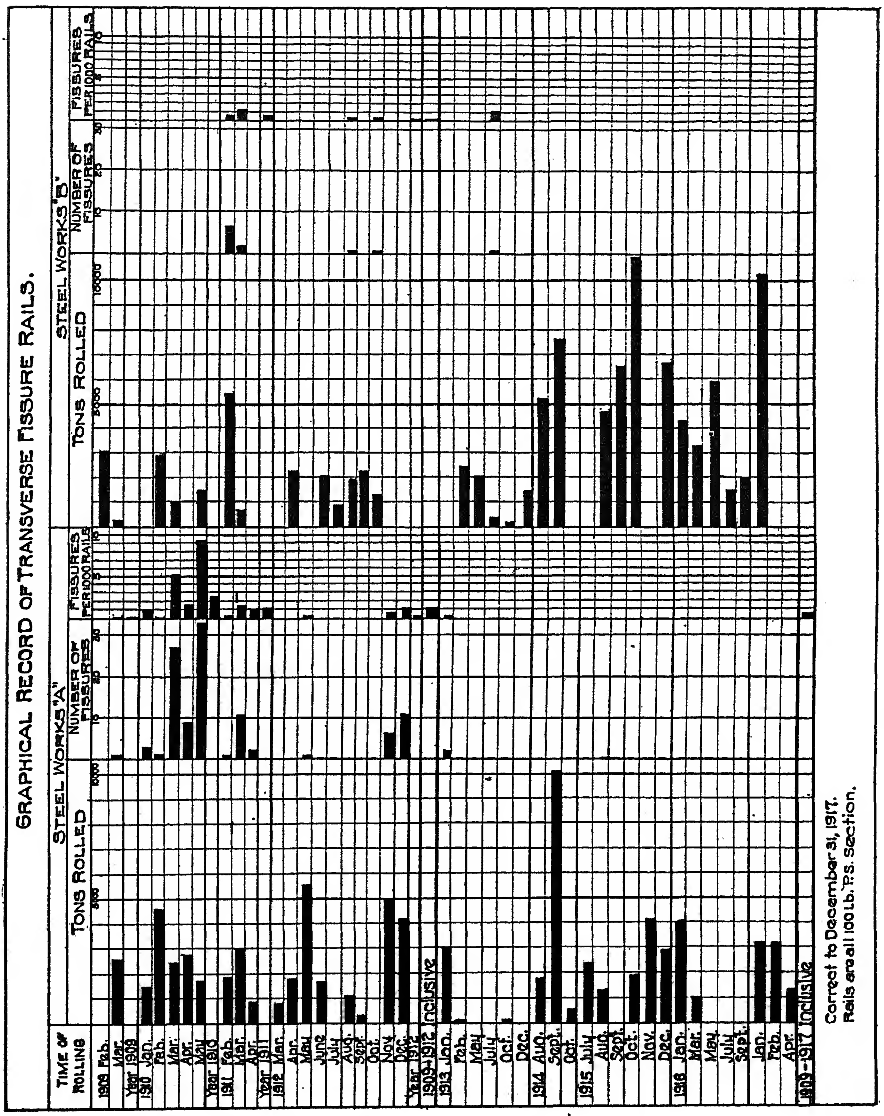
In regard to the rails which have been found with fissures, there is information to show that the material furnished did not meet the specifications in some respects.

During all of this period there has not been brought to my attention a single failure of Bessemer rail with a transverse fissure on the mileage of road, covered by the statistics referred to above, and laid to open-hearth rails.

Mill A furnished, in 1909, 4812 tons, and in 1911, 450 tons, a total of 5262 tons of Bessemer rail. Mill B furnished in 1909, 8544 tons; in 1911, 1500 tons; and in 1913, 6251 tons, a total of 16,295 tons of Bessemer rail. This rail is of the same section and is laid in the same tracks

under the same conditions as the open-hearth rail, and none of this rail has disclosed transverse fissures so far.

A large tonnage of Bessemer rail rolled previous to 1909 is still in the tracks, subjected to the same service as the open-hearth rails, and the




question arises: If transverse fissures are due to fatigue, why do we not have as many failures from this cause in Bessemer rail as in open-hearth rail? This question has not been answered by the mill men nor by Mr. Howard. A mill man's reply to this question was "open-hearth rail

is not ductile enough to stand the heavy loads." It is reasonable to ask, is Bessemer rail more ductile than open-hearth? I do not understand that it is.

Similar questions have been addressed to the two mill companies, as follows:

"Why should there be, comparatively, so many more failures of rails from mill A than from mill B, when rolled to the same specifications, and subjected to as nearly as possible the same service in track?"

"Why should the rails rolled at mill A in May, 1910, prove so much worse than the rails rolled in April of the same year, or so much worse than the rails of March, 1909, from the same mill?"

These questions have not been satisfactorily answered.

There is information in regard to two specific heats of rails rolled at mill A. One heat, rolled in March, 1910, was removed from the track in January, 1916, because of disclosing numerous transverse fissures. Seventy-eight rails from this heat were investigated; 10 disclosed transverse fissures, while the other 68 rails did not. Another heat, rolled in December, 1912, was removed from the track between February and May, 1917; 61 rails were examined, 21 disclosing transverse fissures, and 40 not disclosing them. In the case of this heat, the A rails were rejected at the mill, making this heat a suspicious one. All of these rails were in the track, the rails of each heat being subjected to substantially the same service for the same period of time.

The 78 rails rolled in March, 1910, taken from track, were examined by means of bending tests in a rail-gagging press. One purpose of the examination was to determine to what extent other fissures occurred in this heat of rails, and it was thought that this could best be done by gagging the rails with the head down so as to put this member in tension. The method adopted was to break the rails in two by nicking and bending in the gag press, and then to give each half a bend of about 4 in. Starting at one end, the rail was gagged at short intervals until the other end of the half length was reached, and if the rail stood this test without breaking, the gagging was repeated in the other direction. The amount of the gagging was such as to produce a bend of about 4 in., measured as the distance or ordinate from a string spanning about 16½ ft. of rail, to the middle of its length. When subjected to this test, 58 of the 78 rails stood the bending without breaking. The other 20 rails broke in one or more places, and 10 of them showed transverse fissures, varying in size from some scarcely recognizable to some of large size. In the 10 rails that broke without showing fissures, the fracture radiated from a "gray spot" in the interior of the head, except in the case of rail No. 54, in which one break occurred 29 ft. 7 in. from the head end, the break seeming to start from the top surface; and in the case of rail No. 66, which had a horizontal flaw about ¼ in. from the top of the head and several inches along the side of the head.

A summary of results is given in Table 2.

TABLE 2.—*Summary of Gagging Tests*

| Rail Letter | A | B | C | D | E | F | Total |
|---|---|---|---|---|---|---|---|
| Broken, showing fissures | 1 | 4 | 2 | 3 | 0 | 0 | 10 |
| Broken, without fissures | 2 | 5 | 3 | 0 | 0 | 0 | 10 |
| Total broken | 3 | 9 | 5 | 3 | 0 | 0 | 20 |
| Not broken | 7 | 8 | 6 | 11 | 12 | 14 | 58 |
| Total rails removed | 10 | 17 | 11 | 14 | 12 | 14 | 78 |
| Total in heat | 22 | 22 | 22 | 22 | 22 | 21 | 131 |

One noticeable feature of these results is that there was no breakage of any of the E or F rails, of which 26 of the 43 in the heat were tested. None of the E or F rails broke, and thus, also, no fissures were found in any of them. The highest breakage ratio was found in the B rails and the next highest in the C rails.

The rails that broke in the gagging test and the number of breaks in each rail, both those showing fissures and those not showing fissures, are listed in Table 3.

TABLE 3.—*Breaks in Gagging Tests*

| Rail Number | Rail Letter | Number of Breaks | | Total |
|---|---|---|---|---|
| | | With Fissures | Without Fissures | |
| 19 | C | .... | 3 | 3 |
| 24 | C | .... | 1 | 1 |
| 28 | B | .... | 5 | 5 |
| 32 | B | 3 | .... | 3 |
| 39 | B | .... | 1 | 1 |
| 40 | B | .... | 1 | 1 |
| 42 | B | 8 | 2 | 10 |
| 43 | D | 1 | .... | 1 |
| 45 | A | 1 | ... | 1 |
| 46 | D | 1 | 1 | 2 |
| 47 | B | 2 | 3 | 5 |
| 48 | D | 1 | ... | 1 |
| 49 | B | 1 | .... | 1 |
| 54 | B | .... | 1 | 1 |
| 57 | C | .... | 1 | 1 |
| 62 | C | 1 | .... | 1 |
| 65 | A | ... | 1 | 1 |
| 66 | A | .... | 1 | 1 |
| 70 | B | .. | 1 | 1 |
| 72 | C | 1 | 7 | 8 |
| | | 20 | 29 | 49 |

These breaks are summarized and classified by rail letter in Table 4.

TABLE 4.—*Breaks Summarized*

| Rail Letter | With Fissures | | Without Fissures | | Total | |
|---|---|---|---|---|---|---|
| | Rails | Breaks | Rails | Breaks | Rails | Breaks |
| A | 1 | 1 | 2 | 2 | 3 | 3 |
| B | 4 | 14 | 7 | 14 | 9 | 28 |
| C | 2 | 2 | 4 | 12 | 5 | 14 |
| D | 3 | 3 | 1 | 1 | 3 | 4 |
| Total | 10 | 20 | 14 | 29 | 20 | 49 |

NOTE.—Two B rails, one C rail and one D rail showed breaks, both with and without fissures.

It will be noted that 20 rails broke, with a total of 49 breaks. Twenty of the breaks showed fissures, distributed among 10 rails, and 29 of the breaks were without fissures, distributed among 14 rails; thus 4 of the rails showed breaks both with and without fissures. It is interesting to note that 14 of the 20 fissures occurred in B rails and that 26 of the 29 breaks occurred in B and C rails. One C rail (No. 72) gave 7 breaks, and if we exclude this rail, 14 of the 22 plain breaks occurred in B rails. The A, C and D rails which showed fissures each showed 1 fissure, while the B rails which showed fissures averaged 3½ fissures per rail.

On two of the systems of the Pennsylvania Lines, we had in track Jan. 1, 1917, 219,000 tons of 100-lb. rail and 146,000 tons of 85-lb. rail. Assuming the rails to be 33 ft. long, this would mean about 788,000 rails in the track. The probability is that there is a larger number, by reason of short rails. During the year 1917, there were 451 breakages of these rails, or one to about every 1700 rails. These breakages cover the oldest as well as the latest rails laid. Again, it would appear that there should be a higher ratio of breakages if these rails are much overstressed by their present loads.

It would appear that we have well-founded reasons, from the facts hereinbefore set forth, for the following conclusions:

1. Obviously a large percentage of rails laid should show failure from transverse fissures if these are caused by fatigue due to a repetition of wheel loads.

For example, the figures herewith indicate one failure to about every 3000 rails out of the entire lot, or 1/30 per cent. The other 2999 rails were subjected to the same repetitions of the same wheel loads, and consequently to the same fatigue, and if failure is caused by fatigue, we are clearly justified in expecting a very much larger number of failures

from this cause than we actually find, especially if the material is overstressed, as has been suggested.

It may be said that the above figures are unfair by reason of the fact that a large proportion of the rails had not been in service long enough to develop transverse fissures. To meet this case, let us take, for example, the average of the rails from 1909 to 1912, inclusive, in which case we have one failure to each 1100 rails. This is about 0.1 per cent. We maintain that we should expect to find a higher percentage of failures than this if they are caused by fatigue from overstress.

2. As a corollary from conclusion No. 1, many rollings of rails have not disclosed transverse fissures after having been in service a longer time than rollings of rails which have disclosed transverse fissures. This would indicate that the latter rails must have contained some inherent tendency toward transverse fissures before being placed in the track.

The above discussion may not throw much light on the cause of transverse fissures, but it does seem to indicate that the cause of such fissures cannot be definitely attributed to overstress in the rail caused by the present wheel loads.

F. A. Weymouth,* South Bethlehem, Pa. (written discussion†).—This paper by Mr. Howard summarizes the data that he has collected in his many investigations of rail failures, and should be read in conjunction with the reports of accidents published by the Division of Safety of the Interstate Commerce Commission. The knowledge of the internal strains is not only of great advantage in the study of rail failures but will be very useful in all engineering materials. We have Mr. Howard to thank for the practical determination of these stresses which we *knew* existed, but which we could never, up to this time, determine in concrete amounts.

I think the first questions that come into the minds of those who have read of the determination of these values are: "Is it possible to measure specimens of steel with accuracy to the small fraction of an inch as reported?" and "Is this a *practical* test?" It has been my privilege to work with Mr. Howard on a small portion of this work and I can say that the measurement of these strains to the degree of accuracy reported is a very practical laboratory test, and the work can be duplicated with the same result by any accurate observer with a very little experience in the handling of the instruments.

These conclusions, which form the basis of this paper, show without any doubt whatever that the so-called transverse fissures are absolutely nothing but fatigue failures, which are formed in rails by a combination of internal strains in the rails and the service to which they are subjected.

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* Rail Sales Department, Bethlehem Steel Co.

† Received Feb. 20, 1918.

The obvious, ideal remedy would be to reduce wheel loads, build our roadbeds as permanent rights-of-way with rigid foundations, to eliminate bending moments and vibration as far as possible, and produce rails with no internal strains. But such things cannot be. Economic conditions make it absolutely necessary that wheel loads remain where they are, or possibly be increased in the near future. We cannot look for any radical change in the type of roadbed construction, nor perhaps in condition of maintenance. The internal strains in rolled material cannot be entirely eliminated, but even if this were possible the conditions of their use would introduce strains as soon as the rail is placed in track.

It follows, therefore, that just as soon as we find what the rail has to stand under present average conditions of service and maintenance, we should try to develop a rail that will stand these strains as long as possible. As a means to that end, the Bethlehem Steel Co., in the near future, will build a machine that will test full-sized rails as to their ability to withstand these stresses, which by previous determination we have found to exist in rails under normal service conditions. From a long series of tests of this sort we should obtain valuable information as to the effect of different sections, different composition, and perhaps different methods in the manufacture of rails.

Tests of this character, however, take a long time, and in the meantime this type of failure is constantly on the increase. In what practical way can we cut this down or at least lessen the chances of this type of failure appearing *so soon* after rails are put into service?

A careful survey of all the rails that have failed from transverse fissure brings out the fact that, while they have been found in all types of rails (as is well summarized in the closing paragraph of Mr. Howard's paper) the large majority have occurred in rails possessing extreme chemical hardness. This should not be confused with carbon content, but is better indicated by the Brinell ball or similar test. Rails that are extremely hard have developed transverse fissures earlier in their life and in greater numbers than rails which do not have this extreme hardness. Obviously, it is incumbent upon consumers of rails to stop placing in tracks rails that by all known tests are shown to be extremely hard. While the responsibility of the safety of the tracks rests upon the railroads, the rail manufacturers morally share this responsibility and should refuse to produce rails that are so close to the limit of brittleness.

The railroad engineers have had cause for this desire to obtain rails high in hardening elements, in that increase in wheel loads and speed of trains materially reduced the life of rails on curves. It was to obtain a rail that would give good resistance to abrasion at these particular locations of their tracks that has led them to specify rails having an increased content of hardening elements. The service obtained by these

very hard rails on the curves was satisfactory, and a comparatively few breakages have occurred at these locations, but on tangents where these extremely hard rails were used the breakages have been too frequent. This is unquestionably due to the fact that we are using the same grade of steel for different conditions of use, a procedure that is not found in the use of any other engineering material. In general, we find better maintenance and low speeds on curves, while on tangents we find varying maintenance conditions and maximum speeds. The duty that must be performed by the rails on curves is radically different from that on tangents. The time must come very soon when we shall place in the various parts of the roadway materials best suited for the different service conditions.

A close study of the life of rails on tangents and light curves shows that rails become unfit for service and must be replaced on account of the ends becoming battered or the rails acquiring surface bends and thus giving a rough riding track. Very few rails are ever removed from tangents on account of excessive abrasion. We should, therefore, use for such locations rails with good strength and ductility and of normal hardness, but in no instance should we place under this class of service rails that are close to the limit of brittleness.

Some engineers have insisted on the use of extremely high-carbon rails, having in mind a few trial lots made with very high carbon content but extremely low contents of other hardening elements, which gave good results in service. We must remember, however, that in extremely high-carbon rails containing the usual percentages of other hardening elements we are close to the limit of brittleness.

In one part of Mr. Howard's paper, he says "In the laboratory a steel of about 0.85 carbon displayed the greatest endurance. There are reasons for preferring a lower carbon in rails, and with the lowering of the carbon content a lower limit of endurance would be looked for, other considerations remaining unchanged."

Reference to this is made further on page 603. A finished rail of any section will, as a whole, be a very different engineering member and have different physical properties from a carefully prepared laboratory specimen of the same chemical composition. Rails of any section now in use, made of steel containing over 0.80 carbon, with other elements normal, show in the physical tests a marked tendency toward brittleness and certainly have too low a factor of safety for average service conditions.

There is no evidence to show that a rail of 0.80 carbon or over gives any greater resistance to abrasion on tangents than one 10 points lower in carbon, other elements being equal. Both rails would be removed from service at the same time, but we have an abundance of evidence that rails of the higher carbon have a much greater tendency toward breakage.

At the present time, a few of the railroads are segregating the higher-carbon rails from the others and placing these at points where the service demands a rail that will give good resistance to abrasion. This is a promising step in the right direction. It is very doubtful whether we shall ever *entirely* eliminate the transverse fissure failures, as their occurrence is merely a matter of time, but we certainly can materially reduce their frequency if we keep our rails to a normal hardness—even if we do sacrifice a little of the resistance to abrasion (which I think is very doubtful)—and develop for each particular type of service a grade of steel that will most satisfactorily meet these different conditions.

J. S. Unger,* Pittsburgh, Pa. (written discussion†).—Mr. Howard's reference to the fatigue of metals under repeated stresses is a subject which has not been given the consideration it merits in the study of rails.

As Mr. Howard says, "Brittleness under repeated stresses does not constitute evidence of defective or inferior metal." Ductile metals, such as copper and low-carbon steel, can be made to show brittleness under prolonged alternate stresses, or what is equivalent to brittleness, a sharp break across the section without appreciable elongation. This failure quickly reaches the maximum as the load is increased, as shown in Interstate Commerce Commission Report of Feb. 12, 1917, p. 40.

Compared with the preceding examples, the average rail of, say, 0.65 per cent. carbon and a tensile strength of over 100,000 lb., is not a particularly ductile material, yet it is used under conditions of repeated alternate stresses. If transverse fissures are the result of the combined influence of such stresses and the cold rolling of the head, is it any wonder rails fail from such fissures, especially when the stresses are close to the fiber stress of the steel?

Enough experimental work has been done to justify the statement that a transverse fissure may be produced artificially on any rail at approximately any point in its length and either side or the center of the head as may be desired, and this has been accomplished by subjecting the rail to repeated alternate stresses.

Many opinions have been given as to the cause of these fissures, many being the exact opposite of others, as shown in the list of 25 causes given near the close of Mr. Howard's paper. The quotation at the close of the paper from a Report of the Interstate Commerce Commission leaves little to be added.

The remedies for the prevention of transverse fissures must be apparent to all and must necessarily be one or a combination of the following:

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* Manager, Central Research Bureau, Carnegie Steel Co.

† Received Feb. 19, 1918.

1. Support the rail so thoroughly that injurious alternate stresses cannot exist.

2. Decrease the load, or, what is the same thing, the alternate stresses to such a point as to permit of a reasonable factor of safety common to good engineering practice.

3. Increase the section of the rail until the safe limit of endurance is greater than the stresses to which the rail will be subjected.

P. H. Dudley,* New York, N. Y. (written discussion†).—Mr. Howard introduces his paper with the words: "On Aug. 25, 1911, a rail failed on the Lehigh Valley Railroad." This was the first rail which is known to have failed on any railroad in the United States due to what Mr. Howard calls Transverse Fissures, which have their origin only in the interior of the heads of rails. To provide the trackmen with a proper designation and classification, I have named them "interior transverse fissures."

The failure of rails by the induced interior transverse fissures, though recent, is such a positive indication of needed attention in the manufacture and use of rails that it is necessary to examine briefly the first statements of Mr. Howard in his report on the rail mentioned, as these statements form the basis for his subsequent reports. It is also necessary to state some essential evidence he has overlooked as to the causes of interior transverse fissures and their subsequent development in service.

The rail was a 90-lb., A. S. C. E. section, basic open-hearth steel, 20 per cent. discard, rolled direct from the ingot, Dec. 24, 1909, in winter temperatures, and the red-hot rails would cool upon the hot-beds more rapidly than in the summer. It was the top or "A" rail of the ingot, but piped in the web, in which slag having the following composition was occluded: $SiO_2$, 23.2; $Al_2O_3$,[4] 20.5; CaO, 40.8; MgO, 7.5; not determined,[5] 8.0 per cent.

He reported the chemical composition of several portions of the section, from which I have made an average of the elements: carbon, 0.84; sulphur, 0.032; phosphorus, 0.042, and manganese, 1.20. The specification called for a range of carbon from 0.70 to 0.80 per cent.

The rail was laid Oct. 21, 1910, and failed Aug. 25, 1911, after less than 10 months' service, which is nearly the shortest period known for development and failure.

Quoting further from Mr. Howard's report,[6] p. 14:

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* Consulting Engineer, New York Central Lines.

† Received Feb. 20, 1918.

[4] Comprises $Al_2O_3$, $Fe_2O_3$, $TiO_2$, $P_2O_5$, $Mn_3O_4$.

[5] $H_2O$, $CO_2$, a little coal, perhaps $SO_3$, and alkalies.

[6] Report to the Interstate Commerce Commission on "Accident on the Line of the Lehigh Valley Railroad, near Manchester, N. Y., Aug. 25, 1911."

The cold rolling of the running surface of the head of the rail by heavy wheel pressures has introduced internal strain in the steel at and for a short distance below the running surface. Those internal strains are of compression and they reach their greatest intensity on the gage side of the head where flow of the steel takes place in a lateral direction. That lateral flow is shown by the ridge or fin of metal which was formed along the gage side of the head. When the rail is intact and the top surface or layer of metal of the head is in a state of initial compression, it follows that there must be an opposite strain of tension to balance it in some portion of the rail. The tensile component would be expected in the vicinity of and next below the part which was affected by compression strains, and this leads to an explanation of the presence of fissures in the head where they are being formed. No foreign substance in the steel is needed to account for the inception of those fissures. The introduction of internal strains in a hard and less ductile steel than heretofore used for rails, under excessive wheel pressures, seems adequate to explain the occurrence and frequency of these interior fissures.

The fissures, if this explanation of their origin is true, should not be found in new rails, unused, and it is understood that the steel company that rolled this particular rail has been unsuccessful in finding any such fissures in new rails, notwithstanding a very energetic effort has been made to discover them. These fissures are not held to be defects resulting from improper mill practice. They appear to be the result of excessive wheel pressures acting on a very hard steel, steel which is hard by reason of its chemical composition.

The meager description of Mr. Howard's transverse fissure is stated in his report, page 9, fourth paragraph, second line as follows: "It had a piped web and in addition thereto, it displayed in the head a number of dark spots which should be said are light silvery ones in certain fresh fractures. The surfaces of these spots are normal to the length of the rail."

The etched sections in his report (Nos. 15, 16 and 24) indicate that the impurities in the setting steel of the ingot were large for a 20 per cent. discard. A slight increase of impurities, with traces of slag and alumina in the setting metal of the central portion of the ingot, when rolled out into the section of the rail, often forms, by its junction with the surrounding metal of greater purity, the outline of an inverted pear-shaped mass in the head over the web.

The occluded slag and its composition, the piped web of the rail, and the entrained impurities of the steel, so well outlined by Mr. Howard's etchings, did not suggest to his mind errors of mill practice, as they have to the minds of the engineers of the railroads who have not accepted Mr. Howard's conclusions as to the origin of interior transverse fissures in rail heads. The report discusses in one paragraph the shearing of the blooms, but does not mention:

1. The teeming of the molten steel into ingots.
2. The general practice at that date of throwing metallic aluminum into the molten metal in the molds to make the steel set quiet, except for the New York Central Lines.
3. The dimensions of the ingots.

4. Stripping and the proper time of charging them into the reheating furnaces to equalize their heat for blooming, yet avoid piped ingots.

5. Direct rolling, or from reheated blooms.

6. The rolling of the blooms either by light or heavy roll pressures into the long bar of the section for two or more rail lengths.

7. Rolling an occasional bar so cold as to reduce the ductility of a 16 per cent. chemical composition to 4 or less per cent.

8. Sawing and cambering the rails.

9. The cooling and recalescence of the steel in the section of the rails on the hot-beds.

Several of the constructive processes enumerated above depend upon the temperature of the steel for their accomplishment, and each must be in sequence of proper time intervals. This is required to produce uniform physical properties in the steel rails. A delay in a time interval for the next process or a slip in the intended good mill practice may affect the resulting physical properties in an occasional rail.

The rails on the hot-beds do not cool straight in surface or line, but are either "low" or "high," or a part "low" and another part "high" in the same rail. The "low" or "high" rails must be made straight by the application of the gag under the straightening presses, which puts a permanent set in the metal of the section at each place the gag is applied.

The "low" rails are gagged upon the base to shorten its metal and lengthen that of the head. The "high" rails are gagged upon the head to shorten its metal and lengthen that of the base. The rail in either case is a girder and the metal in the base and head must be adjusted with reference to the length of its neutral surface to be finished straight.

To return for a moment to the Lehigh Valley Railroad piped rail; its inclusions of slag show that something abnormal had occurred in the making of the steel and ingots and in the fabrication of the steel rails; yet Mr. Howard did not even take the precaution to report tensile or other tests of the metal in this failed rail near the center of the head to discover inherent brittleness.

It is easy to determine brittleness due to chemical composition or mechanical defects, but the physical brittleness in commercially sound steel is not so readily found. It is this feature which renders the investigation of interior transverse fissures apparently contradictory and difficult. The subject, however, is of such importance that the technical corps of the railroads will continue until the problems are solved in accordance with the facts.

It is the practice of the New York Central Railroad to subject to drop tests all rails which fail from interior transverse fissures, to determine whether or not the metal of the rail heads on either side of the fracture

is brittle or ductile. The test piece is placed head down on the supports and the tup dropped on the base.

Many of the rails tested under the drop in 1913 and 1914 and later were brittle throughout their entire length, with two or more interior transverse fissures undeveloped. Some rails, except for the short section including the interior transverse fissure, were ductile for the rest of their length, yet the ductile portion carried the same wheel loads as that which failed. It is such positive evidence, confirmed by several other rails from the same melt tested under the same traffic, without failure and without injury to the metal, which proves that some defect must exist in the metal of the rail head in the vicinity of the developed interior transverse fissure.

The specimens in my laboratory, after investigation, I classified by June, 1915, into two general types, *intergranular* and *coalescent*. These were discussed in several technical papers, and require only a brief description here.

*Intergranular Type.*—The nucleus of the intergranular type is checked more or less between the grains and in some cases the crystals are quite perfect, while the fissure which may cause the fracture of the head of the rail develops through the grains from the periphery of the nucleus. This type occurs in rails or parts of rails which cooled "low" on the hot-beds and were then gagged upon the base to shorten its metal and lengthen that of the head.

*Coalescent Type.*—This is a checked longitudinal elliptical nucleus, the imprint of the gag on the non-ductile or weak metal ½ in. or more below the bearing surface of the head, which enlarges to a longitudinal fissure under the passing wheel loads. This fissure, through a serrated crack of the injured metal in the head, finally coalesces—normal to the rail—into the vertical interior transverse fissure. The latter progresses daily from the repeated minor tensile strains of the wheel loads in the wheel spacing and may rupture after several years of service. This type develops in rails or parts of rails which cooled "high" on the hot-beds, and were then gagged upon the head to shorten its metal and lengthen that of the base.

In the failed rails since late in 1915, which broke under the drop test, the fractures are examined under a microscope for crystals and coarse texture. Short pieces from such tests are cut and split for cores, heterogeneous metal, physical or mechanical defects, while heating and cooling curves are made for possible delayed transformations.

The accompanying Fig. 1 shows a rail head which contained an interior transverse fissure of the intergranular type; the nucleus is formed of distinct crystals which can be recognized on the photograph, magnified four diameters. The steel has had sufficient temperature and time for the crystals to grow, and while the work of rolling was sufficient to reduce

those in the surrounding metal, the brittleness of the interior, long overheated metal, cannot be restored. This is by no means an isolated case, for we have several specimens of rail heads containing crystals which cannot be ascribed to the effect of "excessive wheel loads."

Mr. Howard's subsequent reports of rails which failed in the track, by interior transverse fissures, mentions tensile tests from the center of the head in which the ductility is nil, and a piece from the Alford rail, after annealing, indicated a ductility of only 3 per cent.

There is a marked reduction in the number of interior transverse fissures developed in the rails manufactured in the past few years. The two main tracks of the New York Central Railroad between New York

FIG. 1.—INTERIOR TRANSVERSE FISSURE IN RAIL HEAD. INTERGRANULAR TYPE. NUCLEUS, CRYSTALS.

and Chicago contain about 625,000 rails, figured on a 33-ft. basis. We have had 313 rails fail from interior transverse fissures in these two main tracks in the last 5 years. Of these rails, 176 are from three small lots made by direct rolling in November and December, 1910, and January and February, 1911, amounting to about 15,000 tons. Should we subtract these 176 failures from the total, it leaves only 137 breakages during 5 years for the great majority of the rails, most of them made by direct rolling.

The average service of rails in the main line is 8 to 10 years, and the rate of failures for the present larger tonnage will about double the former figure, or, say, 275 for the main line. In 625,000 rails, 275 failures is at the rate of one in 2250, per 10-year period, or one in 22,500 rails per year. Stated another way, this rate is one failure per 7 miles of track per 10-year period, or one failure per 70 miles of track per year. On a percentage basis, this is 0.044 per cent. per 10-year period; yet successful efforts are being made to eliminate this small percentage in future rollings.

Mr. Howard, in his paper now under discussion, page 599, says "concerning the fiber stresses in the track under slowly moving locomotives, observations were made by the writer in 1893 and 1894, the results of which were published in the reports entitled "Tests of Metals."

The last paragraph states, "At that time a rail was selected, representing good track conditions and measured on frozen gravel ballast. It showed a variation in measured strains in different parts of its length of over 100 per cent., referring to the effect of one of the driving wheels. The aggregate longitudinal tension from eight wheels, at one place on the rails, was nearly three times that at another place along its length, while the compressive stresses displayed at one place were 10 times those measured at another place. The measured strains, although taken in track 22 years ago, should effectually dispel any illusions that rails are uniformly strained even under static conditions of loading."

I made those Bessemer rails of 0.06 phosphorus and 0.56 to 0.65 carbon, in 1891 to 1898, when the low-phosphorus ores were exhausted. There were over 64,000 of those 30-ft. rails, including the usual 10 per cent. of shorts, and many of them were still in heavy freight service until the summer of 1917, when they were replaced by stiffer rails, for they were outclassed as girders. Those rails were not "uniformly strained" according to Mr. Howard's opinion, yet, in their long years of service, did not develop a single interior transverse fissure, or what he now erroneously calls a fatigue fracture.

I have spent the parts of several years at the steel mills engaged in the manufacture of rails, and then have followed their service in the track year by year. I have measured the unit strains in the rails under moving locomotives and cars by my stremmatograph and have found some of the favorable relations which can exist between them and the rails in the roadbed. It should be stated that the maximum strains in the base of the rail occur directly under the wheels; also the maximum compression in the head of the rail. At the point of flexure in the rail, these reverse to tension in the head, which may be one-fourth and sometimes one-third of that under the wheels.

The wheel spacing governs the spans of the bending rails rather than the tie spacing, which is not generally understood. I do not find, in the

many tests of the unit strains in rails with my stremmatograph, which autographically records those under the wheels in the base and also in the wheel spacing, that our rails are overstrained.

Personally, as one of the engineers who has had the responsibility of rail manufacture and the investigations of interior transverse fissures, I find so much that Mr. Howard has evidently overlooked that I am unable to accept his conclusions of their origin, or agree that they are fatigue fractures.

G. J. Ray,* Hoboken, N. J. (written discussion†).—In the first instance, Mr. Howard tells us (top of page 598) that the effect of service conditions is the real explanation of transverse fissures, and not the details in the manufacture of the rails.

Again, on page 608, the author makes the following statement: "the statement that a transverse fissure is a modified type of fatigue fracture is based upon the identity of the manifestations which attend and the conditions which lead to rupture in each case, and, furthermore, upon the absence of data leading to other explanation."

For the present, let us assume that the failure is forever and properly named—a modified type of fatigue fracture.

The next question is, what kind of steel rail are we going to purchase in order to be sure that the summation of the various stresses, including those due to the present wheel loads, speed of trains, etc., will not exceed the limit of stress below which, as stated by Mr. Howard, the steel's endurance is practically unlimited? If such steel rails can be manufactured, how are we to make sure that each heat of a rolling, and indeed each ingot of every heat, has the necessary physical properties? Surely, the laboratory test referred to by Dr. Howard cannot be made in all cases, yet from our experience with present mill practice we know how impossible it is to get steel rail of uniform quality rolled under any known specification.

It is my opinion that there is ample evidence to show that mill practice does in some manner or other influence the transverse fissure.

Near the top of page 599, Mr. Howard tells us that, "in the laboratory tests a steel of about 0.85 carbon displayed the greatest endurance." In the same paragraph, we are told that, "with the lowering of the carbon content a lower limit of endurance would be looked for, other considerations remaining unchanged." We have found, in practice, that open-hearth steel rail 0.75 to 0.85 carbon or thereabouts, does not show any flow of metal to speak of under the present wheel loading and heavy traffic. We have also found, to our sorrow, that transverse fissures are most sure to occur in steel rail with carbon above 0.80 and,

* Chief Engineer, D. L. & W. R. R.

† Received Feb. 19, 1918.

in order to reduce the number of failures by transverse fissures, we have very materially lowered the average carbon with correspondingly good results in reducing the number of transverse fissures and in extending the life of the rail before such failures begin to make their appearance. It seems to me this is pretty good evidence that practical results on the track do not in all cases correspond with Mr. Howard's laboratory results.

If the prevailing rail section, or the present track structure is wrong, the manufacturers and other critics have not as yet suggested a practical modification of either to overcome the trouble. At this time, a joint committee of the American Society of Civil Engineers and American Railway Engineering Association, with financial aid from the railroads and the manufacturers, is making extensive tests and gathering much good data, all of which should go far to help decide these questions.

If we have reached the point where the present wheel loading is too great for the steel, regardless of the weight or form of the rail, we are certainly in a deplorable situation.

In the second paragraph, on page 603, Mr. Howard states: "Next, it is important to ascertain the limit of endurance of rails which have been in the track and have acquired a state of internal strain. There is no apparent reason why this barrenness of fundamental data upon steel rails should longer continue." The Lackawanna has lots of data for rail rolled back as far as 1908, and much evidence to show the limit of endurance in actual service, but, of course, we do not know anything about the internal strains acquired by the various rollings except in so far as they have been determined by Mr. Howard in certain samples furnished him for experimental purposes. We do know that certain heats are more susceptible to transverse fissures than others.

In 1913, the Lackawanna had 46 heats of steel rolled at one steel mill during the month of January. Before this rail had been in service a year, transverse fissures had occurred in one heat and it became necessary to remove that heat from the track by the time the rail had been in service two years. Until the spring of 1917, or after four years' service, no other transverse fissures had shown up in any of the remaining 45 heats, although the rail was all laid in the same track. The heat which failed was very low in ductility as compared with the average of the remaining heats.

I have noted many other similar instances to convince me that certain heats are subject to transverse fissures, at least to such an extent that they tend to develop earlier in the life of the rail. Very careful observation of a mass of data has shown conclusively that the per cent. of elongation developed under the drop hammer is uniformly less with the heats in which transverse fissures have developed than with heats in which no transverse fissures have occurred. It is undoubtedly a fact,

as stated by Mr. Howard, that transverse fissures have occurred with rail of all weights and ages. Nevertheless, in my experience with 80-lb. Bessemer rail, rolled at Scranton, we never had a transverse fissure, although much of this rail was still in service and carrying heavy wheel loads and big tonnage up to within the last few years. Not only did transverse fissures never appear, but the rail never did give any trouble from breakage. If such rail could be manufactured in the '90s, why is it that it cannot be manufactured now?

My first experience with the transverse fissure was in 1909, some 2 years prior to the date of the wreck referred to by Mr. Howard. Ever since that time, I have been making an earnest effort to ascertain how to avoid accepting heats of steel in which transverse fissures are liable to occur. The more I study the subject, the more I am convinced that mill practice and chemistry do influence this type of failure. I am also convinced that it is entirely within the range of possibilities to manufacture rail with reasonably good wearing qualities and at the same time not be liable to failure due to transverse fissure, within a reasonable life of the rail, under the present wheel loading and dense traffic.

With one steel mill putting out many thousand tons of rail rolled since 1911, we have had only one transverse fissure, and that was in 101-lb. rail rolled in 1913, after 5 years service.

A. W. Gibbs,* Philadelphia, Pa. (written discussion†).—Having carefully read Mr. Howard's paper, and listened to the discussion at the meeting, I fear that we shall not get far in this matter unless certain proved facts are considered.

It must, I think, be admitted that the service does very greatly affect the development, in the rail, of this as well as other kinds of failures, and that where the greatest density of traffic in combination with the greatest intensity of loading is found, the greatest number of rail failures of all kinds are to be expected.

Mr. Howard's paper does not give proper consideration to the fact that in the same stretch of track, subject to identical service as to tonnage and speed, certain rails show one or a number of transverse fissures, while their neighbors remain sound.

Is it not true, therefore, that it is the service which has the selective property of picking out these rails which are predisposed to failure of this kind? While not claiming that transverse fissures existed originally in the rail, I do claim that certain rails are predisposed, from the time they leave the mill, to the development of this kind of failure when exposed to service.

Unfortunately, it is not practicable to make an investigation of the

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* Chief Mechanical Engineer, The Pennsylvania Railroad Company.

† Received Feb. 21, 1918.

interior condition of any given rail and afterward subject that rail to service, and a post-mortem in case it develops a transverse fissure. Therefore, the best that we have been able to do is to examine representative rails when new and others, associated in the same heats, which have failed from this cause.

I cannot but believe that there is initially a difference in the rails which fail as compared with others in the same heat and service which do not fail, and until our investigations show us what this difference really is, we shall make no real progress toward applying a remedy.

Our records very clearly show that certain rollings have been entirely immune from this kind of failure, while other rollings have shown so large a number of them that it is impossible to believe that it is the service condition which accounts for it. To illustrate: Several years ago in the rolling of our heavy section, two mills each furnished almost exactly 10 per cent. of the rails received for the year. One mill's rail has no failures to date; the other has closely 60 per cent. of all failures for all mills rolling that section that year, and 63 per cent. of the transverse fissures. There have been other instances almost as marked.

It seems to me, therefore, that work such as that instanced by Dr. Dudley, in tracing back to the mill in the effort to identify the cause of the failure, is the only helpful line of progress leading to a remedy.

One speaker suggested three remedies: Perfect track; reducing the weights of equipment; and heavier rail. This is simply begging the question. Perfect track has never existed, and can never be expected. No more can perfect equipment be obtained by any practical maintenance. Weights of equipment cannot be reduced unless we are to retrograde. Heavier rail must justify itself by a marked reduction in failures of all kinds; at present it is on trial.

It must be said that the best rail we are now getting is good enough, judging by service results. On the other hand, we are getting, under the same specifications, rail which we cannot believe is good enough, judging by the results. Our problem is to detect, before failure, what produces this difference and apply a remedy for the bad rail.

W. R. Webster,* Philadelphia, Pa. (written discussion†).—It may be of interest to refer to a paper read by W. C. Cushing, before the American Railway Engineering Association in 1913,[7] "Investigation of Silvery Oval Spots, Sometimes Called Transverse or Internal Fissures in Rail Heads." This was based on examinations made at the Altoona laboratories of the Pennsylvania Railroad. The following is a quotation from Mr. Cushing's paper:

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* Consulting Inspection Engineer.

† Received Mar. 16, 1918.

[7] *Proceedings, American Railway Engineering Association* (1913), **14,** 413.

These examinations have shown that the spot contains slag, which produces a weak place during the manufacture of the rail. It is very likely true, as some have said, that the fracture is developed in service, intimating that extreme severity of service is the main cause for failure. It is clear from these studies that one principal cause of this defect is the inclusion of slag during manufacture, and the subsequent service is only one agent in its development. Indeed, it has been stated positively and authoritatively to the writer that such a spot has been found in a rail which had never been in service. It seems to the writer, therefore, that the assignment of extreme severity of service as the real cause of this type of failure was a mistake, and that it is evident from these examinations that the cause is due to defects of manufacture. It is a fact that the majority of these failures have occurred in the product of one manufacturer.

In discussing this paper, I differed with these conclusions and called attention to a paper on "The Correct Treatment of Steel" read in 1901, by C. H. Ridsdale, before the Iron and Steel Institute of England, from which the following abstracts are taken:[8]

When molten steel cools it crystallizes, the pure iron "grains" settling out, and the more quietly and slowly it cools, the larger they are. The last part to set contains more of the carbon and impurities, and may be termed the "cement" which binds the grains together. If disturbed just as the grains are formed, this cement is still so liquid or soft that they have little or no cohesion, and the material is quite "rotten" or red short in the extreme. At a little lower temperature it becomes cohesive and freely plastic, and it can therefore be readily worked, the cement being so soft that the grains, though cohering enough to permit this, are not held rigidly in their relative positions, but are able to move about each other so easily that they are not themselves appreciably broken up; and if the work is stopped while at this temperature, especially if the cooling is slow, the grain is found to be very large and coarse.

In fact, at this temperature the size and shape of the grain is not affected by work, only by the interference and other conditions of cooling, and the piece exhibits no flow lines and has no rolling hardness. The larger the grain, however, the less coherent it is (owing to the larger area of the cleavage planes) if subjected to sudden shock; so the piece is wanting in toughness and may be actually "rotten."

In discussing Mr. Ridsdale's paper, J. E. Stead stated:[9]

In the tension-testing machine not much difference was observed between a coarse-grain, and a fine-grained crystalline steel when the strain was gradually applied; but under a falling weight the difference was most marked, and often the coarse-grained steel would snap like a carrot. Such fractures were not due to intergranular deposits, but to true separation of the cleavage planes. The large crystal masses present large planes of weakness, and when a strain was brought to bear upon these crystals, they separated through their mass, and once the cleavage was started it traveled rapidly from crystal to crystal through the whole section of the steel. When I was studying, many years ago, the crystalline structure of steel, I obtained very coarse crystalline steel, which elongated 30 per cent. in the testing machine, and yet when a small section was placed upon a V-block, and a sudden blow was given so as to put the under surface in sudden tension, on examination of the piece under the microscope, I found that one or two of the crystals in the center of the piece in which the cleavages happened to be vertical or at right angles to the surface, had fractured.

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[8] *Journal of the Iron and Steel Institute* (No. II, 1901), **60**, 52.

[9] *Op. cit.*, 98.

Is it not possible that the hammer blow of a flat wheel may start fractures in the head of the rail, as described by Mr. Stead, and thus cause a detailed fracture having this silvery oval appearance at the point where the fracture started, due to the surfaces moving slightly on each other before the final break took place; or might they not be started by heavy gagging in straightening?

M. H. WICKHORST, Chicago, Ill.—The literature of this subject practically starts with Mr. Howard's[10] original report on the Manchester wreck in 1911, and although we were all very much puzzled when the fissures were first presented, considerable information has developed since then.

Mr. Howard describes the fissure as a detailed fracture. I think we can agree with him as to the last stages in the production of the fracture, as being due to a detail working, but the simple transverse fissure always starts from a core or nucleus, and the question that has not been settled is how that core forms. It is evidently a sudden rupture all at one time, and after that sudden internal fracture occurs, the rest is a matter of development by detail. This fracture might be called an internal earthquake, or "steelquake."

It is found that these simple transverse fissures (not the longitudinal fissures we have long known as split heads), occur mostly in hard steel, with about 0.80 per cent. carbon; and that the distribution of the chemical elements is pretty uniform. There is no great internal segregation. Therefore, the requirement in some steel rail specifications, that the internal part of the head shall not vary in composition more than a specified percentage from the external part, is not a protection against transverse fissures.

Microscopically, Mr. Howard and others have found that the material at the core is practically the same as the material surrounding it; it seems, therefore, that we cannot find a solution by microscopic methods.

When the rails are examined physically, it is found that the internal metal of the head is of low ductility; that is, when the internal metal is examined, there may be a stretch that, in a good many cases, is not measurable; in other cases it may run up to several per cent., but low ductility of the interior metal of the head is concomitant with all cases of transverse fissures in rails.

As a safeguard against fissures, rails should be tested with the head in tension, either in the drop test or the hydraulic bend test, as it has been found that a rail may give a good result when tested with the base in tension, with defective metal present in the interior of the head. The

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[10] Report to the Interstate Commerce Commission on "Accident on the Line of the Lehigh Valley Railroad, near Manchester, N. Y., Aug. 25, 1911."

usual elongation requirement of 6 per cent. should also be increased to 10 per cent.

ROBERT JOB,* Montreal, Canada.—Mr. Howard has referred to these failures as being the first in the United States. As Mr. Ray mentioned this afternoon, they were not the first in the world; failures of this same type had occurred elsewhere, quite a long time previously.

Several years ago I had some correspondence with Prof. Anton von Dormus, of Austria. It developed that about 1897, he had found a good many of this same type of failure. He published the results of his investigations in Vienna in 1903. Prof. Dormus' photographs show clearly the same type of failure that we find today.

Prof. Dormus mentioned to me in correspondence that the steel in which the fissures occurred ran from 0.20 to as high as 0.40 carbon. The manganese was low. I took occasion to bring this to the attention of the *Railway Age Gazette,* where it was published Feb. 6, 1914, including the photographs from Prof. Dormus' paper. It is interesting reading in connection with some of the statements which have been made today and previously, for here we find steel running as low as 0.20 per cent. carbon containing these fully developed transverse fissures. Steel of that composition was naturally of relatively high ductility. Prof. Dormus also mentioned that whenever that type of failure occurred, the steel was unsound. Under these Austrian conditions we thus find the formation of fissures in unsound rails with low carbon, low manganese and low wheel loads, but *only in unsound steel.*

I remember very well the old type of John Brown rail which we had on our tracks on the Philadelphia & Reading for years; they withstood heavy traffic and were taken up from time to time and replaced in other parts of the track where the conditions were severe. The carbon averaged about 0.35 per cent. We never had a single case of failure of the fissure type, although those rails were in track under extremely heavy conditions of traffic and in small sections, not exceeding 67 lb. per yard.

I also remember well the old Scranton rails. The section of these ranged with us from about 80 lb. per yard to 90 lb., and their composition was about normal for that time, 0.50 to 0.60 carbon, sometimes higher. It was Bessemer steel with phosphorus generally averaging about 0.085 to sometimes as high as 0.090. The size of grain was relatively fine, although much coarser than that of the John Brown rails. Those rails were subjected to heavy traffic and no instance of an internal transverse fissure ever came to my attention.

It is well recognized that a transverse fissure is a result of "fatigue failure," but the same statement is true of many other kinds of failures. The important fact is that fissures occur in very small numbers as com-

* Vice-President, Milton Hersey Co., Ltd.

pared with the total number of rails in service today, and that the vast majority of rails stand up under present traffic conditions without a sign of fissure.

To my mind, the brief statement which I have made in connection with these various types and compositions of rails proves conclusively that the influence of the process of manufacture is a vital point in connection with the service which the steel is to give. In my own practice, examining many thousands of tons of rails, both at the mill and also in track, that is the conclusion which has been forced upon me and from which I cannot escape.

In investigating this particular type of failure, I have repeatedly taken a rail which has failed by transverse fissures, removed it from the track, placed it under the drop, head downward, and struck it with the drop every few feet along the length. In a good many instances the fissured rail would break into many pieces, showing a considerable number of transverse fissures. We would also take up another rail immediately preceding the one which had failed, and one immediately following it, generally from the same heat, the same rolling, and of almost identically the same composition; we would apply the drop (2240 lb. falling 18 ft.) to these every 2 ft. along their length, and bend them most readily without developing a single instance of transverse failure.

Mr. Wickhorst has made some reference to ductility, and Mr. Ray, as I recall, mentioned that some of the rails which developed fissures on his lines, though low in ductility, still had the limits called for by the specifications. I think that we should not forget the fact that 6 per cent. ductility is really extremely low for rail steel of the composition given by the A. R. E. A. specifications for 100-lb. rails, and that normally we can expect to get anywhere from 12 per cent. up to 25 per cent. of ductility, in steel of good average quality—an amount which is far above the low percentage which is given in the ordinary specification today.

J. E. Howard (author's reply to discussion*).—It is gratifying to note that the discussion of the subject of transverse fissures in steel rails has been participated in by so many who are interested in problems pertaining to steel rails. A little over seven years have now elapsed since the report on the accident which occurred on the Lehigh Valley Railroad was published, in which report prominence was given this type of fracture. It appears that divergent views are still held as to the cause or causes which are responsible for their development. The discussion seems to show that the number of ascribed causes has materially diminished during this septenary period. Certain ascribed reasons which were directly opposed to each other have been abandoned as untenable. Attention is now centering, where it should, on the condition

* Received April 15, 1918.

of the metal at the nuclei of the fissures and the stresses which there prevail.

To the author, this type of rupture clearly appeared as a fatigue fracture. Its interior origin, which distinguishes it from the ordinary fatigue fracture, was explained by the presence of metal next the running surface of the head where it is in a state of initial compression. The nuclei of the fissures are located at those elements in the interior of the head which are in a state of initial tension. These initial strains result chiefly from the cold rolling action of the wheels on the top of the head, the metal next the running surface acquiring a state of compression, while the interior of the head, nearer the center, acquires a state of tension, in offering the necessary reaction to the metal which has the opposite sign.

Rails in the track are subjected to repeated alternate stresses, ultimately attaining millions of repetitions. Under such conditions steels fail by fatigue, when the necessary number of repetitions of stresses of the required magnitude have been received. To this statement there are no exceptions. Conditions are present in the track which invariably tend toward the formation of fatigue fractures; when the fiber stresses reach sufficient magnitude and an adequate number of repetitions of loads have been applied, rupture is inevitable. The circumstances which have attended the display of transverse fissures in many cases lead to no other conclusion than that of fracture by fatigue. Critical examinations have been made of the structural, the microscopic, the chemical and the physical states of the steel at the nuclei of the fissures, which have been found to have developed in numbers where no criticism attached to the quality of the steel.

Wheel loads act along the entire length of the rail, introducing a state of compression over its entire length. Transverse fissures display themselves in all parts of the length of the rail. They persistently appear in the greatest numbers on the gage side of the head, a circumstance in itself quite sufficient to excite comment and direct attention to the conditions of service as being primarily responsible for their formation. In fact, as a bit of direct evidence, the predominance of fissures on the gage side of the head is quite sufficient to fix the responsibility on track conditions, even in the absence of knowledge as to the state of the internal strains in the rail which are set up by the action of the wheels.

All grades of steel which have been used for rails acquire internal strains under present wheel pressures. There is no inherent reason why transverse fissures should not develop in rails of any chemical composition; for certain practical reasons, however, they do not develop in all rails. Present wheel loads would cause such distortion of the head of a soft-steel rail that such rails would be removed for other causes

before a transverse fissure developed. In the hard rails, resistance to abrasion is very great. Such rails retain their shapes but acquire internal strains, and without offering visible evidence of their presence. It is not unlikely that the retention of shape promotes the formation of transverse fissures since the elements most strained remain the same ones. If the breaking of a piece of wire is undertaken by repeated bending, labor is lost by shifting the bending stresses to a new place along its length from time to time. The same would be expected to hold true in shifting the maximum stresses to a new part of the cross-section of the rail. It presents no inconsistency to have rails develop transverse fissures without material change in the shapes of their heads. In fact, lateral flow of the metal of the head, causing the formation of fins, affords relief to the longitudinal strains.

While the author has undertaken to point out definite reasons for the formation of tranverse fissures, as a modified type of fracture by fatigue, based upon the identity of conditions which are common in the service of rails and other examples of fatigue fractures, it is somewhat disappointing to find so few criticisms from those who have kindly participated in the discussion which suggest tangible fields for the further examination of rails which display this type of rupture. The author is engaged in exploring those features and phenomena which intervene between the period when the rails are in possession of their primitive properties, undisturbed by service conditions, and the time when rupture ensues.

The phases through which steel passes from its primitive state of integrity to its final rupture has been given very scant attention. There has been no lack of tests on the primitive state of the metal. The relations between chemical composition, methods of treatment, and those properties which must inevitably follow in accordance with natural laws, should be well established by this time, and possibly admit of modification in existing methods of test. Tests for acceptance, however desirable they may be, nevertheless constitute but a limited part of the information, much of which is of vital importance, which is demanded upon those vicissitudes which are encountered subsequently when the rails are put into service. It is highly desirable to acquire exact information upon all phases through which the metal passes and determine with accuracy the effects of conditions of service upon the integrity of the metal of the rails.

In respect to the criticisms which have kindly been made upon the paper, Mr. Gennet remarked upon the desirability of examining the metal in the vicinity of the nuclei of the fissures, which of course was very early done. He mentions the progressive character of the development of the fissures, their growth being due to track conditions, his views being in accord with those which are entertained by the author. Mr.

Gennet thinks the examinations have not progressed sufficiently, however, to absolve the steel from responsibility for the origin of the fissures, in respect to chemical composition and physical condition, fearing also that interior conditions may be present, at the place of rupture and at other places, manifold and minute, requiring careful and skillful work in the examination by trained metallurgists and microscopic observers. During the 7 years which have elapsed since this type of fracture was brought into prominence, it is probable that nearly every laboratory in the country has done some work on this subject. It is to be regretted that the testimony of some of the railroad laboratories, in which such investigations have been made, was not presented among the criticisms. So far as known, however, the results obtained tended to confirm the views expressed by the author. It is recognized that a certain degree of vagueness pertains in the application of metallographic results to the endurance of steel under repeated stresses. The relations between microstructure and the ability of steel to endure long-continued stresses remain to be established. It is not always clear what constitutes a micro-defect in steel, according to different observers. The most critical statement that has reached the author was made by a British metallurgist in connection with the St. Neots accident on the London & Northwestern Railway, which occasioned a Parliamentary investigation. That authority recognized the existence of some 34,000,000 micro-defects per cubic inch of steel. Early efforts made by the author to acquire microscopic evidence upon the effect of millions of repeated alternate stresses, which had ended in the rupture of the steel, did not result in the identification of a microscopic change in the steel. It is not unreasonable to infer that some of the phenomena observed in severely overstrained steel, by means of microscopic examination, may constitute evidence of an advanced stage in the disintegration of the metal. Such evidence, however, has to do with the effects of previous overstraining, and does not have reference to the primitive state of the steel. The crucial microscopic test is one which will indicate the safe fiber stresses which may be applied to the steel, or which will indicate the number of repetitions of load of given magnitude which may safely be applied, or points out some structural or chemical condition, the modification of which will effect an improvement in the durability of the rail, together with such information as will enable this modification to be made in the process of manufacture. It is needless to say that such ends have not yet been reached.

Mr. Gennet refers to current specifications governing the acceptance of rails. Specifications have been revised from time to time. It is recalled that an early authority wrote that "steel rails of the best quality may be bent double when cold, or twisted an entire turn in a length of 2 ft. without cracking." Rails are now being accepted which will dis-

play only 5 per cent. elongation. Railroad officials have been complimented for accepting, and putting into service, rails of extreme chemical hardness, having more than 1 per cent. carbon content. Resistance against abrasion and wear has been made the criterion of excellence. The trend of thought is already swerving in the direction of specifying rails which have greater primitive toughness than required by current specifications. Compliance with any set of specifications does not carry with it immunity from ultimate rupture of the rails. The critical feature is the determination of the ability of the steel to endure such stresses as the rails are subjected to in the track. Multiplying the number of primitive tests, and going no farther, does not solve the problem of what constitutes safety in the track. It is thought to be an error to attach such reverence to current specifications as to believe their fulfillment will be the means of arresting the formation of transverse fissures. Such a belief overlooks the basic difference between the display of physical properties when steel is tested to destruction, in its primitive condition, and its ability to endure long-continued stresses, none of which exceeds or even closely approaches its primitive elastic limit. When this fundamental difference is appreciated the subject of transverse fissures will be much better understood.

Mr. Isaacs believed that the display of transverse fissures in rails which show very little wear controverts the explanation that they are a type of fatigue fracture. On the contrary, that fact is one of the reasons which clearly establish it as a fatigue fracture, and as such has been referred to by the author. Mere increase in weight, as Mr. Isaacs states, has not brought immunity from transverse fissures. Increase in weight is accompanied by an increase in the section modulus and diminution in the magnitude of the fiber stresses in the rail considered as a beam, but increase in weight has not brought about a substantial amelioration in the cold-rolling effects of the wheels on the head. One of the components which is believed to be responsible for the development of transverse fissures remains practically unchanged. The early display of fissures in rails of the heavier sections attaches a considerable share of the responsibility to the effect of the internal strains in promoting this type of rupture. It is important to ascertain the relative influence of the cold-rolling strains of the wheels and the direct bending stresses from the same source. The behavior of the heavier sections in this respect is significant. It has long been known that the negative bending stresses in the head of the rail are lower than the positive bending stresses in the base. Mere increase in weight of rail should lower each of these values. On the other hand, features of interest are the intensity of the cold-rolling strains in the head, the volume of metal which acquires a state of compression, and the rigidity of the section in its influence on the reacting strains of tension in the interior of the head. Upon these features investigations

of the author are yielding definite values, the results being made available in current publications.

Mr. Isaacs has prepared a table of hypothetical cases of loading, extending that introduced by the author up to a range of repeated stresses of 80,000 lb. per square inch, in order apparently to point out that the range in stresses is greatest at the surface. It is not necessary, however, to prepare a table to show that the greatest range in stresses is at the most remote fibers from the neutral axis, since this feature is covered in early mechanics. The point at issue concerns the location of the longitudinal elements which are exposed to the greatest strains of tension. They are located in the interior of the head in the vicinity of the nuclei of transverse fissures.

Transverse fissures developed by progressive gagging, and gagging in one plane which formed a part of the series of tests, threw light upon three interesting features. Gagging in one plane ultimately fractured the rails but caused fractures which had their origins at the surface. Progressive gagging through its distributive influence affected the entire length of the rail, resulting in the formation of transverse fissures. These fissures were located central over the web, on the right or on the left side of the head, according to the manner in which the gagging blows were directed. The number of gagging blows required, per rail length, ranging from 50,000 to 250,000, showed the negligible influence of the 10 to 15 blows in the gagging press required for the usual straightening at the mill, each of the latter being less severe than those employed in the progressive gagging tests.

Mr. Isaacs remarks still further that "The relation that exists between wheel loads and track stresses has not been even approximately determined." The paper presented did not enter upon this aspect of the rail question. Some tests were made upon track stresses by the author in 1893 and thereafter, as stated in the body of the paper. It is recognized that some questions pertaining to these stresses are indeterminate, but additional data are becoming available through recent tests, inaugurated, it is understood, by the American Society of Civil Engineers.

Mr. Isaacs remarks also that "transverse fissures are more prevalent in the product of some mills than in those of others," to which might be added the statement that transverse fissures are also more prevalent on some railroads than on others. Carrying the analysis farther, it might be shown that they are more prevalent on some divisions of the same road than on other divisions, and appear more frequently in the low rails than in the high rails of curves. In the main, the density of traffic seems to be the controlling factor, those rails which are most strained developing transverse fissures. Since the product of some mills is largely taken by roads on which traffic is dense, it is not surprising that a difference in the number of fissures should be noticed, quite inde-

pendent of the chemical specifications to which the different mills may have been required to work. Time is required to develop transverse fissures, and the time interval is reduced on those roads where traffic is heaviest. It has been stated that from 2 to 3 years are commonly required on some roads to develop fissured rails. It has been said that rails possessing primitive inherent defects are culled out quite promptly after they reach the track, quite the reverse of experience in the development of transverse fissures which have been reported to display themselves after long periods of service, or to increase in number following the introduction of heavier equipment.

Mr. Isaacs adopts the views "that a limit cannot be put upon the fiber stresses in rails in track until the relation of wheel loads and track stresses has been established beyond a doubt;" and further, "that reliance must be placed upon failed-rail statistics to show where caution is necessary and where reasonable safety is provided." It is true that a stress limit can not be put upon material concerning which the stresses are unknown, but I do not agree that safety and reliance should rest on statistics based upon the failure of rails in the track. It seems dangerous to adopt such a method of acquiring information for such a purpose, attended as it is with a menace to life and destruction of property.

The internal stresses which reside in rails, amounting to thousands of pounds, appear to have been given no consideration. The endurance of rails under different fiber stresses in their primitive state, in full cross-section of the rails, has not been made the subject of inquiry. The residual strength to endure repeated alternate stresses after the rails have been for a time in service, and have acquired high states of internal strains, remains undetermined. The customary drop tests for acceptance furnish no conclusive information upon these important matters. Apparently no efforts have been made to acquire data on these matters, although this could readily be done. The laboratory, and not the track, is believed to be the place to acquire certain fundamental data upon rails in defining factors of safety. These remarks are intended in no way to suggest any cessation in road-bed and track tests and observations.

Mr. Trimble presents interesting data, confined to track statistics, showing the prevalence of transverse fissures in cases which have come before him. He enumerates many interesting queries, which track data do not solve. With the excellent resources at his command, it is to be regretted that it was not feasible to furnish more detailed and specific information upon track conditions, and supplement those data with the results of laboratory tests and examinations on the rails which failed.

One recalls the remarks of Mr. Benjamin Talbot, the distinguished English metallurgist, who said concerning rail failures, "Great interest on this subject has been aroused in America, due no doubt to the ac-

cidents which take place on American railways, which, fortunately, do not occur in Great Britain. One explanation was that in Great Britain the rails were not treated in the barbarous fashion customary in the United States." It will be conceded that track records on the failure of rails admit of substantial advance before the limit of improvement is reached in respect to furnishing exact and reliable information. The annual statistics of rail failures emphasize the need of acquiring more exact information, but they do not fully supply the need. Mr. Trimble's observations were confined to open-hearth rails. The experience of other engineers has included both open-hearth and bessemer rails. Mr. Trimble summarizes in the following words: "The above discussion may not throw much light on the cause of transverse fissures, but it does seem to indicate that the cause of such fissures cannot be definitely attributed to overstress in the rail caused by the present wheel loads." If there were no overstressing of the rails the thousands of pounds of internal stresses would not be introduced in them, and the rails after a term of service in the track would remain with only the primitive strains which were acquired during fabrication.

Dr. Dudley recognizes that "failure of rails by the induced interior transverse fissures, although recent, is such a positive indication of needed attention in the manufacture and use of rails that it is necessary to examine briefly the first statements of Mr. Howard in his report on the rail mentioned," etc. It is known that the manufacturers have had this matter under consideration during the last 7 years, and their efforts have been reënforced by the researches conducted under the direct auspices of the Committee on Rail of the American Railway Engineering Association. The manufacturing end of the subject has apparently received more direct attention than the use of the rails in the track, and is presumably farther advanced, if it is proper so to speak of a subject which has yielded so many negative results. The net result seems to be that no chemical, structural or microscopic reason has been discovered which can be held responsible for the development of transverse fissures. Published remarks are recalled in which Dr. Dudley has announced "that interior transverse fissures in rail heads are induced by a combination of two or more exceptional conditions of manufacture in an occasional rail head by direct rolling, which can and should be avoided." So far as known, this is the only direct statement which has been made of the discovery of conditions of manufacture which are responsible for the formation of transverse fissures. If this discovery is confirmed, an improvement in conditions will be expected as soon as remedial steps have been taken which eliminate these occasional and exceptional conditions of manufacture.

Dr. Dudley notes the omission from the paper of references to "the teeming of the ingots, the use of metallic aluminum, the dimensions of

the ingots, the stripping of the ingots, reheated blooms, light or heavy roll pressures, occasional cold rolling reducing the normal ductility 75 per cent., sawing and cambering, the cooling and recalescence of the steel." It has not appeared to the author that these features have the direct bearing on the formation of transverse fissures which stresses and strains have. Since some of these matters are ordinary and not exceptional conditions of manufacture it would seem to follow also that they were negligible factors according to Dr. Dudley's views.

There still remains, according to the remarks of Dr. Dudley, "needed attention in * * * the use of rails," a remark in which the author concurs, to which may be added the desirability of continuing investigations upon the phases through which the rails pass under the influence of track conditions.

Mr. Ray very properly raises the question as to what kind of steel rails can be procured which will not be influenced by the various stresses and conditions enumerated by the author, and held to be responsible for the formation of transverse fissures. This is preëminently a practical question and the one above all others to which it would be gratifying to furnish a complete and satisfactory answer. To the author there is but one way in which rupture by repeated stresses can be avoided, which, of course, is to remove the material from service before the number of repeated stresses have reached a number approaching the ultimate endurance of the material. This answer to Mr. Ray's very pertinent question is not so vague as may seem upon first sight. There is a limit to the endurance of all grades of steel. That limit of endurance admits of ready definition in the case of laboratory tests. It also admits of definition in the case of rails in full cross-section. The primitive endurance of rails can be determined and also their residual strength after periods of service.

The state of the strains in rails is obviously very complicated, both before and after the rails have been placed in the track. It defies our efforts to ascertain the exact relations which exist between them in promoting or demoting endurance of repeated stresses. The safe criterion, and at present the only one, consists in subjecting full-section rails to repeated stresses.

Practically all grades of steel which can be held suitable for rails have been tried. No grade has successfully endured the conditions of service in the sense in which steel is used in permanent structures, omitting consideration of abrasion and wear. Increase in sectional dimensions is going on. It is conceivable that a section, not impracticable in its weight, may be reached which will render rails immune from the formation of transverse fissures. Such a size apparently has not yet been reached. Alternate bending stresses alone do not cause transverse fissures. They will cause rupture, in which the outside fibers

are first to fail. Cold rolling has such a tendency, but examples of interior fissures in the absence of concomitant bending stresses have not been witnessed by the author.

If increase in section accomplishes the elimination of transverse fissures it will be due to a reduction in the severity of the combined bending stresses and internal strains. It must be admitted, however, that the case does not look very promising. There is no escape from the local impinging effects of the wheels on the running surface of the head of the rail. Increase of section effects an improvement in the bending stresses without correspondingly ameliorating the local impinging effects of the wheel pressures.

Whether or not unity of opinion is reached in regard to the causes of transverse fissures, the summation of all relevant influences is told in the formation of such fissures in the track. If it is found that a state of saturation of internal strain is reached, if the changes in physical properties in respect to the relations and values of the elastic limits in tension and compression reach a stable condition, and repetitions of loads then remain as the only variable factor, it will afford opportunity to ascertain the residual strength in rails withdrawn from the track for testing purpose and enable judgment to be formed of the approach to rupture. The answer to Mr. Ray's question is found in such tests and examinations of rails as will show the residual endurance in possession of the rail after a term of service.

Rails must be used under present equipment. If their life is limited, they must be removed from service and new rails put in their places. A transverse fissure is a type of fracture which raises the query whether wheel loads are not approaching such a maximum that it is unwarrantable further to increase them.

# Slag Viscosity Tables for Blast-furnace Work*

BY A. L. FEILD,† A. B., M. S., AND P. H. ROYSTER,‡ A. B., A. M., PITTSBURGH, PA

(New York Meeting, February, 1918)

THE first report on the slag viscosity work of the Bureau of Mines was made by one of the authors[1] in 1916. It was concerned chiefly with the method of measurement. A paper[2] on this phase of the work was presented to the Institute in February, 1917, and a similar paper[3] was communicated to the Faraday Society.

## PURPOSE AND SCOPE OF THE SLAG VISCOSITY TABLES

Following the development of a suitable method, the next step in the investigation was the measurement of slag viscosity over a wide range of temperature and composition, and the arrangement of this experimental data in the form of tables for the use of the blast-furnace operator. It is believed that this information, if used intelligently, should help the operator to reduce losses in production caused by off-grade pig iron, to improve fuel economy, to promote operating efficiency, and to extend present-day practice to meet the increasing need of smelting lean and complex ores.

In the experimental part of the work it was early observed that the temperature-viscosity relations of a slag were determined, for practical purposes, by the relative proportions of the three major constituents, lime ($CaO$), alumina ($Al_2O_3$), and silica ($SiO_2$), provided the percentage

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[1] A. L. Feild: A Method for Measuring the Viscosity of Blast-Furnace Slag at High Temperatures. *Bureau of Mines Technical Paper* 157 (1916), 29 pp.

[2] A. L. Feild: The Viscosity of Blast-Furnace Slag. *Trans.* (1916), **56**, 633.

[3] A. L. Feild: The Viscosity of Blast-Furnace Slag and Its Relation to Iron Metallurgy, Including a Description of a New Method of Measuring Slag Viscosity at High Temperatures (communicated by Sir Robert Hadfield, F. R. S.). *Transactions of the Faraday Society* (1917), **13**, Pt. I.

of none of the minor constituents, CaS, MgO, MnO, FeO, $TiO_2$, $K_2O$, and $Na_2O$, was abnormally high. Since magnesia, MgO, is the most important of the minor constituents and is apt to vary considerably in different slags, its effect was investigated in greater detail. It was concluded that the temperature-viscosity relations of slags containing not more than about 8 per cent. MgO were, for practical purposes, determined only by the relative percentages of CaO, $Al_2O_3$, and $SiO_2$.

The problem then resolved itself into a measurement of the viscosities of synthetic slags containing lime, alumina, and silica over a range of composition corresponding to commercial practice. On the concentration-temperature diagram of the system $CaO-Al_2O_3-SiO_2$, as determined by Rankin and Wright, the area covered by the tables lies partly in the calcium metasilicate and partly in the gehlenite stability field. For each even percentage of $Al_2O_3$ from 7 to 24 per cent., two tabulations are given. One shows the temperatures at which the slag attains a viscosity of 2, 4, 6, 8, 10, and 12, for varying proportions of CaO and $SiO_2$; the other shows the viscosities of these slags at 1350°, 1400°, 1450°, 1500°, 1550°, and 1600° C.

Before using the tables in practice, the slag composition as shown by analysis must be calculated to a basis of 100 per cent. CaO, $Al_2O_3$, and $SiO_2$.

## Conclusions from Results of Investigation

The strictly scientific side of the investigation is discussed in another Bureau of Mines report.[4]

The most important conclusion to be drawn, however, is that, even when a slag is melted and completely converted into a liquid, it is composed of relatively complex compounds of lime, alumina, and silica, instead of a simple mixture or solution of the constituent oxides. The temperature-viscosity relations of a slag are not, therefore, changed continuously and in the same general direction by the gradual addition of a single constituent. While this discovery will ultimately necessitate a revision of our common ideas regarding the chemical properties of slags, it does not in any way enter into the practical use of the tables, where values are referred to the relative amounts of lime, alumina, and silica as determined by chemical analysis.

## The Function of the Slag and Its Dependence on the Temperature-viscosity Relations

The composition of the slag determines the temperature of the smelting zone at a given rate of driving. It also determines the temperature-

[4] A. L. Feild and P. H. Royster: Temperature-viscosity Relations in the Ternary System $CaO-Al_2O_3-SiO_2$. *Bureau of Mines Technical Paper* 189. Abstract in this volume.

viscosity relations of the slag and thus directly the length of time during which the iron is in intimate contact with the slag. This intimate contact between iron and slag takes place only when the slag is formed and continues until the slag becomes fluid enough to permit the iron globules to fall out of the slag to the bottom of the hearth. Therefore, the length of time of contact is not determined by the viscosity of the slag at any one temperature, but by the rate of change of viscosity with temperature. A slag which shows a wide range of temperature from the pasty stage, which insures initial contact, to the stage at which the iron is permitted to separate and drop to the bottom of the hearth, will give a long contact period. A slag which changes in viscosity rapidly over a shorter range of temperature will permit of a relatively shorter time of contact.

The reactions of desulphurization and of silica reduction, while occurring simultaneously, are of a distinctly different nature. When these reactions begin to occur appreciably, the sulphur is for the most part in the pig iron. The process which takes place has more or less the characteristics of a distribution of the sulphur between the pig iron and slag. The velocity of this distribution is limited by the viscosity of the slag, and gradually becomes smaller and smaller as the percentage of the sulphur in the iron decreases. On the other hand, there is at the beginning no silicon present in the pig iron. It is all present as silica in the slag. This silica, when in contact with molten pig iron and carbon at high temperatures, is reduced to metallic silicon and enters the pig iron. This reaction never approaches completion and hence is determined almost entirely by the time of contact and the temperature. The actual viscosity of the slag probably affects the extent of reduction only slightly, except in so far as the viscosity determines the time of contact. In other words, a fluid slag is not necessary for the process, since diffusion is a minor item.

Briefly, then, the following general statements may be made:

1. A high temperature favors a high silicon and a low sulphur content in pig iron.

2. A long period of contact favors both a high silicon and a low sulphur content.

3. A moderately long period of contact, at a moderate temperature, will produce a pig iron low in silicon and with a reasonably low sulphur content.

4. A short period of contact, at a high temperature, will also produce a pig iron low in silicon and with a reasonably low sulphur content.

5. A long period of contact, at a high temperature, will produce a pig iron high in silicon, and low in sulphur.

Conditions (3) and (4) represent those which prevail in making basic iron by the two familiar methods, respectively, of using either a

moderately basic slag and a heavy ore-burden or using a very basic slag and a light ore-burden. Condition (5) is fulfilled in making foundry iron.

Such statements as the above are not specific. They are qualitative, not quantitative. In order to derive the maximum benefit from the tables, the operator should keep an accurate record each day of the slag composition in the case of each furnace, the weight of slag produced each day, the silicon and sulphur content of the pig iron, the coke consumption per ton of pig, and the pig production; and over against this data should be recorded the temperature-viscosity relations of the slag as shown in the tables.[5] The authors are collecting such data and information.

However, in the case of such a complex problem it is obvious that the focussing of many minds on the same subject and from different points of view will do much toward elucidating the operating problems of the blast furnace in a practical, quantitative manner.

## The Relation Between the Size of the Molten Iron Globules and the Percentage by Weight of These Globules in the Slag

A scientific laboratory study of the reactions of desulphurization and of silica reduction is not possible without some definite knowledge regarding the size of the iron globules which are in contact with the molten slag prior to reaching the iron bath at the bottom of the hearth. A calculation of their average size can be made in any particular case if the daily iron and slag production, the dimensions of the hearth, the slag temperature, and the percentage of iron globules in the slag are known by making use of the tables. In a particular case, such a calculation showed that the average diameter of the iron globules was equal to approximately 0.1 in. (0.25 cm.).

It is not entirely impossible that the radius of the iron globules may be somewhat of the nature of a constant quantity for all furnaces and over a wide range of operating conditions. If this be true, the loss of metallic iron in the slag should be directly proportional to the viscosity of the slag at the hearth temperature and to the rate of pig production.

It is highly desirable to determine the actual percentage of metallic iron in slags of known composition in the case of a number of furnaces.

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[5] In the case of those plants where the chemical analyses of slags has included the actual percentage of lime, CaO, present, as distinct from MgO and other "bases," such a procedure as that suggested above could be adopted to cover a wide range of past practice, from which many valuable conclusions might be drawn. Our experience has been that such information is unfortunately not available at the majority of plants in this country.

### *The Average Slag Composition Used in the United States*

Computation based on slag analyses representing 44 per cent. of the total pig-iron production of the United States shows that the *average* slag, so far as pig-iron tonnage is concerned, is as follows:

| | Per Cent |
|---|---|
| CaO | 40.17 |
| $Al_2O_3$ | 13.07 |
| $SiO_2$ | 35.32 |
| MgO | 5.51 |
| CaS | 3.87 |
| Other oxides | 2 06 |
| | 100 00 |

## Deductions from Data Given in Tables

It is possible to utilize the viscosity values given in the tables in an examination of the validity of the common "rules" for slag practice, in application to actual or hypothetical furnace conditions, and in numerous other ways which cannot be entered into fully here.

For example, the data show that if the alumina content (13.07 per cent.) of the average slag is permitted to increase up to 22 per cent., the ratio of "Bases" to $SiO_2$ being maintained constant, the temperature at which the slag reaches a viscosity of 10 gradually rises from 1352° to 1519° C., and the temperature at which it reaches a viscosity of 4 rises from 1456° to a temperature higher than 1640° C. This shows in a striking manner that a normal high-alumina slag should more readily yield a high-silicon foundry iron than the average slag, because of the higher temperature induced in the smelting zone by the former. On increasing the alumina content, under the conditions stated, from 13.07 to 15 per cent., it is probable that any tendency to produce high-silicon pig iron must be caused by the rise in the smelting temperature alone, while on increasing the alumina content above 15 per cent., both a higher smelting temperature and a longer time of contact between molten iron and slag will tend to increase the silicon in the pig iron.[6] These conclusions are

[6] The slag is in intimate contact with the molten pig iron from the time when the slag is just fluid enough to entrap the molten iron globules until the time when the slag has such a low viscosity that the iron globules fall out of the slag into the bottom of the hearth. Both of these conditions should correspond to a more or less definite viscosity, which, for lack of more definite information, we may place at a viscosity value of 10 in the former case and at a viscosity of 4 in the latter case Assuming a constant rate of heating in the case of the slag, this time of contact should be roughly proportional to the difference between the temperature at which the slag reaches a viscosity of 10 and the temperature at which the slag reaches a viscosity of 4.

in line with the view of J. E. Johnson, Jr.[7] and J. H. Frantz that high alumina might be of benefit in making foundry iron.

In a manner similar to the above, the question of the effect of magnesia on slag viscosity (in amounts up to 8 per cent.) is discussed in the original paper, of which this is an abstract; also, the practice of a constant ratio of "Bases" to "Acids" is briefly considered there.

## The Effect of Varying the Amount of Lime Charged

Suppose we have a furnace using 2000 lb. of coke (910 kg.) per ton of pig, including scrap, the composition of the ore mixture, coke, and stone being as follows:

| Ore Mixture | Per Cent. | Coke | Per Cent | Limestone | Per Cent. |
|---|---|---|---|---|---|
| $SiO_2$ | 7.55 | $SiO_2$ | 4 97 | $SiO_2$ | 3.33 |
| $Al_2O_3$ | 1.96 | $Al_2O_3$ | 3.24 | $Al_2O_3$ | 1.28 |
| CaO | 0.44 | | | CaO | 48.19 |
| Fe | 47 00 | | | | |

Calculation, the details of which are given in *Technical Paper* 187, shows that the slag, produced when 800, 900, 1000 and 1100 lb. of limestone are charged, will exhibit the properties given in Table 1.

Table 1.—*Temperatures Giving Stated Viscosities*

| Limestone Charged per Ton of Pig, Pounds | Viscosity = 10 | Viscosity = 4 | Difference Degrees |
|---|---|---|---|
| 800 | 1,404° C. | 1,526° C. | 130 |
| 900 | 1,389 | 1,504 | 115 |
| 1,000 | 1,376 | 1,479 | 103 |
| 1,100 | 1,478 | 1,612 | 134 |

Table 1 shows in a clear manner to what a marked extent the temperature-viscosity relations will vary when different amounts of limestone are used with any particular ore mixture and coke. Either 800 or 1100 lb. of limestone would probably produce a slag which would be favorable toward the production of a smelting zone of high temperature. On the other hand, it is obvious that 1000 lb. of limestone would be productive of a maximum fuel economy, and would yield a very desirable slag, which would be probably very suitable for basic iron. The use of 1100 lb. of limestone would place the slag very close to the composition where the furnace would tend to become "limed-up" or "lime-cold." In spite of the high lime content of this slag, a small drop in temperature

[7] *Metallurgical and Chemical Engineering* (Apr. 1, 1916), **14**, 367.

caused by irregularities in charge or wind distribution would yield a high-sulphur pig iron. The smallest lime charge cited, 800 lb., might give very satisfactory conditions for foundry iron.

## Applying Viscosity Data to Operating Problems

An examination of the viscosity tables will disclose the fact that the viscosity relations of slags corresponding to certain compositions are changed to a remarkable degree by a relatively slight change in its composition. Every effort should therefore be made by the operator to secure the most representative sample of slag possible under the conditions existing at the plant, and to assure himself that the slag is properly analyzed. A rather extended investigation has shown that (1) in average practice even a fairly representative sample of slag is seldom secured; and that (2) the usual works method of slag analysis, in addition to furnishing incomplete records of slag composition, are liable to surprisingly large errors in the case of slags moderately high in alumina.

The slag viscosity tables cannot be correctly used unless the percentage of $SiO_2$, $Al_2O_3$, CaO, and sulphur are determined *as such* separately. It is suggested that the sodium carbonate fusion method be used in all cases where the alumina is present in amounts above 12 per cent. Analyses are given showing that the hydrochloric acid digestion method, when used on a slag containing about 23 per cent. alumina, gave figures for alumina that were from 6 to 10 per cent. too low, *i.e.*, varying between 14 and 17 per cent. approximately.

The sample for analysis should be taken from a composite sample made up of several slag samples taken from each flush and from the slag at cast. A tentative method of sampling the slag is described. Frequent sampling is necessary because of the wide variation in the composition of the slag from flush to flush and even during the same flush. Analyses are given showing this variation in composition in the case of a furnace on basic iron and a furnace on foundry iron.

## Slag Viscosity Tables

In order to illustrate the general form of the tabulated data, there are given below the data for $Al_2O_3 = 14$. *Technical Paper* 187 contains similar data for every even percentage of alumina from 7 to 24 inclusive.

It sometimes happens that a slag will contain, for a given alumina content, a larger percentage of lime (or lower percentage of silica) than is included in the tables. This, however, is seldom the case, and is a strong indication that the slag is too high in lime for satisfactory furnace operation. In such a case, the slag composition on the ternary diagram

lies in the field of calcium orthosilicate, to which it did not seem advisable to extend the investigation.

A careful study of the authors' discussion as given in the original paper should make it possible for any operator to enlarge upon the topics treated in whatever manner may be best suited to the particular conditions which he encounters.

FROM TABLE 8.—(*Bureau of Mines Technical Paper* 187)

$Al_2O_3$ = 14 (Calculated so that $CaO + Al_2O_3 + SiO_2$ = 100 Per Cent.)

| $SiO_2$ | CaO | Viscosity | | | | | |
|---|---|---|---|---|---|---|---|
| | | 2 | 4 | 6 | 8 | 10 | 12 |
| 48 | 38 | >1,640° | 1,529° | 1,470° | 1,437° | 1,413° | 1,396°C. |
| 47 | 39 | >1,640 | 1,519 | 1,460 | 1,427 | 1,403 | 1,387 |
| 46 | 40 | >1,640 | 1,511 | 1,450 | 1,417 | 1,394 | 1,379 |
| 45 | 41 | 1,638 | 1,502 | 1,440 | 1,408 | 1,385 | 1,369 |
| 44 | 42 | 1,627 | 1,492 | 1,431 | 1,397 | 1,375 | 1,358 |
| 43 | 43 | 1,618 | 1,480 | 1,419 | 1,386 | 1,362 | 1,347 |
| 42 | 44 | 1,604 | 1,466 | 1,405 | 1,373 | 1,352 | 1,338 |
| 41 | 45 | 1,586 | 1,454 | 1,395 | 1,363 | 1,343 | 1,330 |
| 40 | 46 | 1,554 | 1,440 | 1,396 | 1,375 | 1,360 | 1,352 |
| 39 | 47 | 1,635 | 1,498 | 1,438 | 1,406 | 1,386 | 1,374 |
| 38 | 48 | >1,640 | 1,618 | 1,535 | 1,486 | 1,455 | 1,432 |
| 37 | 49 | >1,640 | >1,640 | >1,640 | >1,640 | >1,640 | 1,616 |

FROM TABLE 9.—(*Bureau of Mines Technical Paper* 187)

$Al_2O_3$ = 14 (Calculated so that $CaO + Al_2O_3 + SiO_2$ = 100 Per Cent.)

| $SiO_2$ | CaO | Viscosity | | | | | |
|---|---|---|---|---|---|---|---|
| | | 1350° | 1400° | 1450° | 1500° | 1550° | 1600° |
| 48 | 38 | 21.5 | 11.3 | 7.1 | 4.8 | 3.5 | 2.7 |
| 47 | 39 | 19.2 | 10.2 | 6.5 | 4.5 | 3.4 | 2.6 |
| 46 | 40 | 17.6 | 9.4 | 6.0 | 4.3 | 3.2 | 2.5 |
| 45 | 41 | 15.2 | 8.6 | 5.6 | 4.0 | 3.1 | 2.4 |
| 44 | 42 | 13.4 | 7.8 | 5.2 | 3.8 | 2.9 | 2.3 |
| 43 | 43 | 11.6 | 7.1 | 4.8 | 3.6 | 2.8 | 2.2 |
| 42 | 44 | 10.3 | 6.4 | 4.4 | 3.3 | 2.6 | 2.0 |
| 41 | 45 | 9.2 | 5.7 | 4.1 | 3.1 | 2.4 | 1.9 |
| 40 | 46 | 12.3 | 5.9 | 3.7 | 2.6 | 2.0 | 1.6 |
| 39 | 47 | 13.6 | 8.8 | 5.5 | 3.9 | 3.0 | 2.4 |
| 38 | 48 | 40.2 | 16.7 | 10.3 | 7.3 | 5.5 | 4.3 |
| 37 | 49 | 200.0 | 83.0 | 38.5 | 23.8 | 17.5 | 13.3 |

# Temperature-viscosity Relations in the Ternary System $CaO-Al_2O_3-SiO_2$*

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(New York Meeting, February, 1918)

*Bureau of Mines Technical Paper* 189 consists of a record of the scientific data obtained in the iron blast-furnace slag investigation which is reported in *Technical Paper* 187, "Slag Viscosity Tables for Blast-furnace Work." It includes a description of the method of preparing the synthetic melts, construction of an improved electric furnace, and refinements in laboratory methods of measurement. Only the general features of the work and the conclusions drawn from the experimental data can be given attention here.

## Silicates in the Liquid State

In the manufacture of refractories, Portland cement, glass, and porcelain, and in pyrometallurgical operations, importance attaches to the behavior of silicate compounds at temperatures beyond the range of common experience. This field of research has not been investigated as carefully as its scientific and industrial importance would seem to warrant.

Probably the most important of the unsolved problems in the chemistry of silicates is the question of their stability in the liquid state. That little progress toward the solution of this problem has been made is not strange if there be borne in mind the difficulties attending experiments made at temperatures at which silicates become fluid, and the comparatively short time since it became possible to measure these temperatures. On the other hand, considerable progress has been made in the study of the compounds in the solid state because of the directness and simplicity of methods of optical examination.

Important advances have been made by the Geophysical Laboratory at Washington in recent years. An extended study has been made there of saturated solutions representing a considerable number of binary and ternary systems of the more common refractory oxides. The methods

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employed, however, confined the results to a determination of the identity of the solid phases which crystallized from the melt. From an investigation thus limited, it is impossible conclusively to show that the observed compounds were stable in the liquid state.

The evidence given by Sosman in support of the view that certain silicate compounds are subject to purely thermal dissociation is not conclusive. It has been shown by Bowen that the system $CaSiO_3$ (pseudowollastonite)—$MgSiO_3$ (clino-enstatite) is not a true binary system, but is a part of the ternary system lime-magnesia-silica, since in the greater portion of the curve between diopside and clino-enstatite the primary phase separating is forsterite ($Mg_2SiO_4$) and the residual liquid is a pyroxene.

It can be said, then, that there is no conclusive evidence that thermal dissociation into constituent oxides is a general characteristic of silicates at temperatures above their melting point.

## The Viscosity of Silicates

The phenomena accompanying the change of state of silicates from solid to liquid are shared by few substances. It is a general rule that the physical properties of a body are subject to a discontinuity at its melting point. This rule does not hold for the silicates. The definition of a solid given by Maxwell is that its viscosity be infinite. Kelvin defines as a solid a body capable permanently of resisting a shear, however small. The definition of a solid thus depending upon the physical property of viscosity, it is evident that the final determination of the melting point of a body will alone involve a determination of its viscosity. Where a property other than viscosity—optical, thermal, electrical—suffers discontinuity at the melting point, this may be used for convenience, indirectly, to determine the melting temperature.

Those silicates whose viscosities have been measured show an increase of viscosity with diminishing temperature. The greater number of them appear to increase continuously in viscosity, which becomes infinite asymptotically to some definite temperature. This temperature must therefore by definition be called the melting point. In the case of a few of the silicates, the viscosities of which are here reported, the usual rule for other substances is followed, *i.e.*, the viscosity becomes infinite discontinuously at its melting point. However, the general rule is the case first mentioned.

## Viscosity and Plasticity

For nearly all the silicate mixtures examined, a residual torque was observed when the outer cylinder of the viscosity apparatus, containing the silicate melt, was brought to rest, which was greater than could be attributed to a possible experimental error. This phenomenon of a sub-

stance possessing rigidity under an applied shear below a certain limiting value has been long recognized for those bodies variously termed malleable, ductile, and plastic "solids." Their behavior differs from that of the silicate mixtures alone in that the order of magnitude of the rigidity modulus, the viscosity, and the limiting shear, is much greater.

In order, therefore, to include the above substances, it will be necessary to modify the ideal assumption of a viscosity independent of shear to take account of the known rigidity at low shears. This is treated mathematically in the original paper.

## Relation Between Viscosity, Temperature, and Composition in the Ternary System $CaO-Al_2O_3-SiO_2$

The viscosity of a silicate within the ternary system $CaO-Al_2O_3-SiO_2$ is given[1] by the expression

$$\eta = f\,(L, A, T)$$

where $T$ is the temperature, $L$ the percentage of lime (CaO), and $A$ the percentage of alumina ($Al_2O_3$). The experimental problem, therefore, was to determine the value of this function. The result of these experiments would be, say, a family of surfaces in three dimensions giving $\eta$ as a function of $L$ and $A$ with $T$ equal to a constant parameter for each surface. Synthetic melts were made of appropriate values of $L$ and $A$, and viscosities measured from near their melting point to an upper limit of 1600° C. A construction of these surfaces was attempted only for those portions of the fields of calcium metasilicate and gehlenite lying between 7 and 24 per cent. $Al_2O_3$. These surfaces are of no simple nature, and it did not appear practicable to determine them with any accuracy without knowledge of the general relations to be expected. Experimental measurements on a relatively few compositions outside of the selected field, chosen with reference to the stability diagram, were sufficient, when combined with the measurements made in this field, to permit the construction of the surfaces with satisfactory accuracy.

## Experimental Results

*Technical Paper* 189, of which the present article is an abstract, contains the following experimental data, in the form of viscosity-composition and temperature-viscosity curves:

1. Profiles of isothermal viscosity surfaces along the calcium metasilicate-gehlenite binary system.

[1] The observed rigidity at low shears would demand the introduction of a fifth variable into this expression. Since its effect, however, is not such as would modify essentially the general relations considered, the further complication would not appear necessary at this point.

2. Viscosities in the binary system calcium metasilicate-gehlenite at the saturation temperature and at 50° and 100° superheats.

3. Temperature-viscosity curves for the quintuple points 2, 5, 6, 7, 8, 14, and 20 (as defined by Rankin and Wright).

4. Profiles of isothermal viscosity surfaces along the calcium metasilicate-anorthite boundary curve.

5. Profiles of isothermal viscosity surfaces along the calcium metasilicate-gehlenite boundary curve.

6. Profiles of isothermal viscosity surfaces along the gehlenite-anorthite boundary curve.

7. Profiles of isothermal viscosity surfaces along the calcium orthosilicate-gehlenite boundary curve.

8. Profiles along the 1450° C. isothermal viscosity surface at 7 and 10 per cent. alumina.

9. Profiles along the 1450° C. isothermal viscosity surface at 15, 18, and 21 per cent. alumina.

10. Temperature-viscosity curves of the binary eutectics between (*a*) $CS$–$CAS_2$, (*b*) $CAS_2$–$C_2AS$, (*c*) $CS$–$C_2AS$, (*d*) $C_2S$–$C_2AS$, and (*e*) $C_3S_2$–$CS$ (where $C = CaO$, $A = Al_2O_3$, and $S = SiO_2$).

## Crystallization in the Ternary System

As has been stated above, the viscosity of silicates usually increases continuously with decreasing temperature, and becomes infinite asymptotically to the melting temperature. It would appear, then, that crystallization in a ternary eutectic could proceed with great difficulty, if at all. If crystallization does not occur, it would not be possible to find more than two solid phases in any given melt within the system.

Of the three ternary eutectics (points 12, 14 and 15) in which the presence of three crystalline phases has been observed by Wright, two (points 6 and 14) were among the seven quintuple points on which viscosity measurements were made. These alone do not obey the usual rule of possessing a temperature-viscosity curve which is continuous at the melting point. The existence of a discontinuity in the case of these two curves shows that there would not be an infinite viscosity of the liquid at the melting point, and hence crystallization should proceed without any great difficulty.

Until further evidence is adduced, it is not possible conclusively to state that, within the meaning of the phase-rule, complete equilibrium is ever attained by purely thermal means in the case of a typical silicate.

## Melting Point

It is impossible logically to refer to the saturation temperature of a mixture as a "melting point." In the first place, the crystal which forms

has undergone a change, not from the liquid to the solid state, but from solution to the solid state, *i.e.*, precipitation. The term "melting point" has been so long defined as the temperature at which a solid is in equilibrium with a liquid of its own composition, that it has never seemed wise to extend its meaning to cover the case of saturation.

Aside from saturation temperature, the only temperature which can be associated with the term "melting point" within a ternary system, for instance, will be the temperature at the stable quintuple point. This agrees with the physical definition of melting point, as already given, and there seems to be no sound objection to its use in this sense.

If the temperature-viscosity relations of the compositions corresponding to the quintuple points be represented in terms of fluidity upon the temperature-fluidity diagram, the resulting curve will in each case be more or less a straight line, convex, however, to the temperature axis, fluidity increasing with increasing temperature. Within the range of measurement, it is possible to express the temperature as a power series of the fluidity. In the case of point 2, it is found that the expression

$$T = 1167 + 737\left(\frac{\phi}{100}\right) - 445\left(\frac{\phi}{100}\right)^2.$$

agrees with the experimental values with a mean deviation of 5.5° C. over a range of viscosity from 100 to 2000. If it be assumed that the expression holds for a range of fluidity 5 per cent. greater, that is, as far as $\phi = 0$, the temperature corresponding to zero fluidity would be 1167° C. The melting temperature given by Rankin and Wright is 1170° ± 5°.

Similarly, in the case of point 5, the expression

$$T = 1269 + 132\left(\frac{\phi}{100}\right) - 38\left(\frac{\phi}{100}\right)^2 + 4.9\left(\frac{\phi}{100}\right)^3$$

agrees with the observed values with a mean deviation of 4.4° C. over a range of viscosity from 33 to 200. For point 7, the equation

$$T = 1388 + 35.5\left(\frac{\phi}{100}\right) - 2.9\left(\frac{\phi}{100}\right)^2$$

agrees with the observations with a mean deviation of 1.6° C. over a range of viscosity from 33 to 166. The temperatures corresponding to zero fluidity in these two cases are, respectively, 1269° and 1388° C. Values of melting temperatures obta ned by Rankin and Wright, are 1265° ± 5° and 1380° ± 5°.

## Conclusions

The results obtained are sufficient to point strongly to the conclusion that by measurements of the viscosity of these silicates in the liquid state alone, it would be possible to establish the existence of the compounds

within the system, and to locate more or less closely their fields of stability. The viscosity surfaces, to the extent to which they have been investigated here, show a certain number of maxima and minima, the former occurring at the quintuple points and the latter at the binary eutectics. Knowing that the boundary line between two fields of stability (except in the case of compounds which are unstable at their melting points) terminates at quintuple points and includes one and only one binary eutectic, these boundary curves may be drawn to pass through the observed maxima and minima. It is not suggested that this method of determining the stability fields would be a convenient or even practicable one. But interest in this point attaches to the following conclusion: if it is possible from the measurement of a physical property of a liquid to determine the field of stability of a compound it follows inevitably that the compound exists as such in the liquid state.

It appears that in any qualitative statement regarding the relation between viscosity and oxide composition, this must be limited to the general statement that the viscosity, at such analogous points as binary and ternary eutectics, in the various stability fields decreases with increasing lime, and that the temperature coefficient of viscosity is greater for high alumina content. It is possible that the same effect might be noted in comparing the viscosity of the various compounds, and possibly of similarly situated points on the binary systems. This general tendency toward lower viscosities for higher lime contents at analogous points is much less in magnitude, however, than the effects which are caused by the relative amounts of the compounds present in the liquid melt.

The most immediate effect of the conclusion that the melted silicate is a mixture not of oxides but of somewhat more complex compounds is to raise the question of whether it is correct to consider the oxide as the active constituent entering into the various reactions known to take place in these melts. For example, it has been usual to write the reaction for the desulphurization of iron in contact with blast-furnace slag

$$CaO + FeS + C \leftrightarrows CaS + Fe + CO$$

Although it is permissible to assume that the lime compounds are subject to a certain amount of thermal dissociation, it must be remembered that evidence as to the degree of the dissociation is entirely lacking. If it is desired to write the hypothetical dissociation, say, of gehlenite as

$$Ca_2Al_2SiO_7 \leftrightarrows CaSiO_3 + CaAl_2O_4$$
$$CaSiO_3 \leftrightarrows CaO + SiO_2$$
$$CaAl_2O_4 \leftrightarrows CaO + Al_2O_3$$

it is certainly true that the final distribution of sulphur between slag and the iron bath will depend upon the equilibrium constants in the four equations above. We are forced, then, to conclude that the desulphuriz-

ing power of a slag, as dependent on its composition, cannot be expressed simply in terms of its percentage of CaO.

When an attempt is made to visualize the intimate nature of the processes taking place in the smelting of metallic ores, it becomes necessary to solve certain problems in the flow of viscous liquids. It was with the purpose of establishing the physical constants required in the consequent fluodynamical equations that the viscosity measurements described in this paper were made. These constants are, of course, no less necessary to the study of the formation of igneous rocks. It is seen, however, that conclusions must be drawn from the results obtained which extend beyond the limits of these rather simple industrial and geological problems. They concern in a most direct manner the little-studied chemistry of the silicates.

## Discussion

D. J. Demorest,* Washington, D. C. (written discussion†).—This paper is a real contribution to technical science; it will make it easier to think accurately about the inner workings of a blast furnace. I was rather surprised to read that the authors regarded as a new discovery the idea that a molten mixture of CaO, $Al_2O_3$, and $SiO_2$ is a molten solution of the compounds of these oxides rather than of the oxides in each other. Surely no chemist with an active respect for the known facts of chemical affinity could believe that these oxides could be melted together without forming compounds, which compounds would then dissolve in one another. This work does, however, help to establish what it was supposed must be true.

I think it is worth while, in the light of the work of the Geophysics Laboratory, this work of the Bureau of Mines, and that of previous investigators, to visualize what happens as a charge passes down through a blast furnace, thinking especially of the slag-forming materials. These are chiefly kaolin ($Al_2O_3.2SiO_2.2H_2O$), quartz, and limestone. These substances do not undergo much change until the limestone decomposes at about the middle of the furnace, whereupon the chemical affinities of CaO and $SiO_2$ must begin to be active, especially if there is any amorphous $SiO_2$ present. This forms $CaSiO_3$, and the $SiO_2$, $CaSiO_2$ and $Al_2O_3 2SiO_2$ then come together to form the binary and ternary eutectics so well located by the work of the Geophysics Laboratory. No matter how much lime has been charged into the furnace, the formation of this eutectic will start at about the same height in the furnace and the gangue will start to get pasty at the same height. If the charge is such that the gangue materials are in proper proportion to be entirely consumed in

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† Received Feb. 2, 1918.

making this eutectic the slag will be completely liquid high in the furnace, will soon become superheated, and will run rapidly through the furnace.

This will mean, of course, that the slag will not remain long enough in contact with the coke and reduced iron to remove the sulphur from the iron or to permit the silicon to be reduced. Its time of contact will be short and its temperature will be low. If, however, there is an excess of lime over that required to form the eutectic, this lime will dissolve in the eutectic, make it viscous, and keep it viscous as long as there is lime left to go into solution. It will, therefore, not become free-running until it has gone further down the furnace than the first slag. The more lime present, the longer it takes to dissolve it (I am referring to ordinary coke practice which requires a basic slag), the longer the sticky slag stays in contact with the iron and coke, the hotter it gets, and the longer the time afforded to reduce silicon and remove sulphur. The upper limit will, of course, be a mixture containing more lime than the slag will dissolve; whereupon the furnace "limes up." There is, of course, nothing new in this. It was very clearly set forth in Howe's "Iron, Steel, and Other Alloys." I have none of my notes or books here and cannot write accurately, but I think that these "Viscosity Tables" will correspond well with the triaxial diagram worked out by the Geophysics Laboratory.

F. H. Willcox,* Chicago, Ill. (written discussion†).—Until about six years ago, one of the most striking tendencies in American blast-furnace practice was the use of large amounts of lime. This has, in the past, undoubtedly been carried to an extreme. The difficulties encountered in such practice have, perhaps, overemphasized, in some cases, the significance that may be attached to slag characteristics as a corollary of furnace economy and regularity.

Eleven years ago, when many furnaces in the Pittsburgh district were running decidedly limy, the real reason that they were doing so may have been the fact that the relatively rich ores were actually deficient in slag-making constituents. Gravel was often regularly added to the burden in order to give a volume of slag that would eliminate sulphur satisfactorily. The ores were frequently deficient in aluminous constituents, and some intangible requisite being lacking in the slag, lime was made to do the duty of the missing alumina. It apparently required an inordinately large amount of limestone to do the work that could be done by a considerably less proportion of alumina.

In this same district, plants that then used to run on a total acids of 46 to 47 per cent., with alumina at 13 to 14 per cent., are now running

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† Received Feb. 5, 1918.

on total acids of 51 to 53 per cent., with alumina of 17 to 21 per cent., and the work is very much smoother.

Coinciding with the changes in slag, improvements have been made in lines and distribution. Nevertheless, it is the opinion of some operators that replacement of limestone by clayey and feldspathic gangue in the ore has been of material benefit in their practice.

While Pittsburgh, in these earlier years, was getting the richer ores, to avoid the freight haul on the higher-gangue ores, the Lake ports were taking a higher proportion of the leaner ore. It was at these Lake port furnaces that the use of sharp cinder on basic iron first became established practice. This resulted from the fact that the ores contained an adequate amount and variety of gangue components, just as a limy cinder in the Pittsburgh district followed from the consumption of rich ores. Similar mention might be made of the conditions of furnace practice where slags unduly rich in magnesia or alumina have been carried.

I have made mention of these instances for three reasons: (1) I believe that the character of the cinder made in any given furnace district is very largely governed by geographical situation, and by a variety of relations in the purchase of ore, coke, and limestone such as inter-company associations, traffic conditions, competitive prices, etc., with little attention to the metallurgical significance of the resulting slag. (2) I believe that aside from extremes in basicity, magnesia, or alumina contents, there is a very considerable range over which normal economical practice is obtained. (3) It is my impression that the occasional emphasis placed upon slag constitution, as governing to a marked degree the smoothness of furnace operation, is founded on experience with slag unduly rich in bases, magnesia, or alumina, rather than upon definite data with widely divergent slags afforded by normal practice in the several districts.

Prior to this investigation, there was a smaller amount of exact scientific knowledge of this subject—the constitution and viscosity of fluid blast-furnace slags and their effect upon furnace practice—than upon any subject connected with blast-furnace work. The paper is valuable in that it helps to remove another element of uncertainty from the problems encountered in practice.

The blast-furnace process is still perhaps the most sensitive and the most crude of large-scale manufactures. The chemist, metallurgist, physicist, and engineer have done much toward removing uncertainties in recognizing and treating symptoms, and in eliminating their causes. Efficient stoves, correct lines, good distribution, stronger construction and adequate machinery have largely contributed to past developments, together with the accumulated experience of furnace men in adapting their practice and equipment to leaner ores and changing fuels.

Contributions which advance our knowledge as to the fundamental physical-chemical laws of blast-furnace metallurgy are strongly to be encouraged, and it is to be hoped that the Bureau of Mines will continue to follow investigations in this field.

A. L. FEILD, Cleveland, O. (written discussion*).—I wish to call Professor Demorest's attention to the fact that in the paper under discussion it is shown that there is close agreement between the viscosity measurements and the equilibrium diagram of Rankin and Wright for the ternary system $CaO-Al_2O_3-SiO_2$, in so far as the location of several ternary eutectics are concerned, and that this agreement extends to the boundary lines of the various stability fields. A certain emphasis, however, appears to be needed here as to the relative industrial applicability of the determinations of the saturation temperatures of this system at the Geophysical Laboratory, and the measurements of viscosity within the system here presented. From the former data information is available as to the melting points of the eutectics and compounds, and of the mutual solubility relations within the diagram. However—and this point should be borne in mind—a knowledge of the melting point of a eutectic alone gives no information whatsoever as to its viscosity-temperature relations. There are certain eutectics belonging to this ternary system which possess such a high viscosity throughout the range of blast-furnace temperatures that it would be impossible to operate with them, from purely mechanical considerations; and these extremely vscous eutectics are, in some cases, possessed of a very low melting point. On the contrary, a viscosity-temperature curve gives, not only the melting point of a silicate mixture in the physical and mechanical meaning of the word, but furnishes definite information concerning the viscosity at all temperatures, and hence sufficient data for a calculation of the slag flow under all possible imposed conditions. The difficulty in arriving at precise answers to all but the more simple cases of hydrodynamic flow is not inherent in the viscosity data but in the field of pure mathematics, *i.e.*, in the complexity of the differential equations involved. Approximate solutions, sufficiently accurate for commercial use, can probably be obtained with the exercise of a little ingenuity.

Mr. Willcox's discussion certainly contains valuable material, especially from the historical standpoint. Viewed largely, it is difficult to regard the blast furnace and the operations progressing within it as of a very delicate nature, in the strict meaning of the word. The delicate point is the matter of the proper decision as to how to handle operating irregularities. If we exclude explosions and breakouts, a blast furnace continues to operate after a fashion even in the most distressing circum-

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* Received April 8, 1918.

stances. Of course, the pig iron may be off-grade or not the proper iron. An accurate analysis of the blast-furnace process would undoubtedly have been made previously if perfectly definite slag composition and slag volume were required. It is the fact that the smelting of iron is a process which occurs with satisfactory results over a wide range of operating conditions, combined with the fact that there has not been available reliable information as to the location of this range of operating conditions in terms of slag composition and volume, burden, rate of driving, and blast-temperature, which has been responsible for the seeming delicacy of the process.

# Grain-size Inheritance in Iron and Carbon Steel

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(New York Meeting, February, 1918)

THIS paper will include a brief discussion of Prof. Howe's paper on The Supposed Reversal of Inheritance of Ferrite Grain Size from that of Austenite.[1] The general subject of grain refining in steel and iron will also be treated.

The conclusions reached are stated in the following hypotheses which are discussed more in detail below.

1. The ferrite grain size in pure iron, the ferrite and pearlite grain size in hypoeutectoid steel, the pearlite grain size in eutectoid steel and the cementite and pearlite grain size of hypereutectoid steel are not inherited from the grain size of the mother austenite.

2. The only structural feature that is generally inherited from the austenite of hypo- and hypereutectoid steels on cooling through their transformation ranges is the position of the excess ferrite or cementite at the austenite grain boundaries, sometimes causing complete and sometimes incomplete networks which outline the old austenite grain boundaries. Rapid cooling through the transformation range will cause the non-inheritance of this structural (network) feature.

3. The austenite grain boundaries themselves are nearly always effaced in all steels and also in pure iron during the *Ar* transformations.

4. The grain-size refining of steel and iron is brought about by the combined effect of non-inheritance of the transformation products on either heating or cooling, *i.e.*, the austenite transformation products do not inherit their grain size from the austenite on cooling through the transformation range nor does austenite inherit its grain size from the products which form austenite on heating.

5. In general, in both iron and carbon steel the larger the austenite grain size, the larger will be the grain size of the transformation products on cooling. This, of course, assumes all other conditions constant except the austenite grain size. An exception is found to this general rule in very pure iron such as electrolytic iron. In this instance small austenite grains may form very large ferrite grains on cooling through $Ar_3$.

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[1] This volume, p 487.

6. In iron and steel, the larger the ferrite, cementite or pearlite grain size, the larger will be the austenite grain size on heating above the $Ac$ transformations, and *vice versa.*

7. The faster the rate of cooling of iron and steel through the $Ar$ transformation range, the smaller will be the grain size of the transformation products and *vice versa*.

8. The faster the rate of heating of iron and steel, other conditions remaining the same, the smaller will be the austenite grain size.

9. The greater the temperature gradient during the transformations in iron and steel on heating or cooling, the larger will be the grain size.

10. If the grain size of a transformation product in iron and steel immediately after the transformation is smaller than the equilibrium grain size of that product under the existing conditions, the equilibrium grain size will be established by the known laws of grain growth.

11. A single grain of any constituent in iron or steel (austenite, ferrite, pearlite) when forced by thermal treatment to undergo one of the $A$ transformations, must transform from at least one nucleus, but may and nearly always does transform from more than one nucleus.

## Discussion of Hypotheses

The grain size of any transformation product in iron and carbon steel due to heating or cooling through the transformation range will depend on:

*A*. The rate of heating or cooling through the transformation range.

*B*. The grain size and arrangement of the constituents just prior to the occurrence of any transformation.

*C*. The chemical and physical composition of the iron or steel product, *i.e.*, carbon content, sonims, gas inclusions, blow holes, etc.

*D*. The temperature gradient obtaining during the transformation period.

The manner in which these general laws influence the grain size in the transformation products of iron and steel can be illustrated by citing some actual examples.

### 0.7 *Per Cent. Carbon Steel*

A piece of 0.7 per cent. C steel was heated to a temperature of 1100° C., held for 5 min. at that temperature and then cooled in the furnace. The structure of this steel is shown in Fig. 1 at a magnification of 100 diameters and in Fig. 2 at a magnification of 400 diameters.

Since the ferrite network marks the boundaries of the old austenite grains, it is found by grain-size measurements that just prior to the $Ar_1$ transformation there were 36 grains of austenite per square millimeter. The ferrite and pearlite grain size, however, vary greatly from the number of austenite grains. There are 250 grains of ferrite per square millimeter

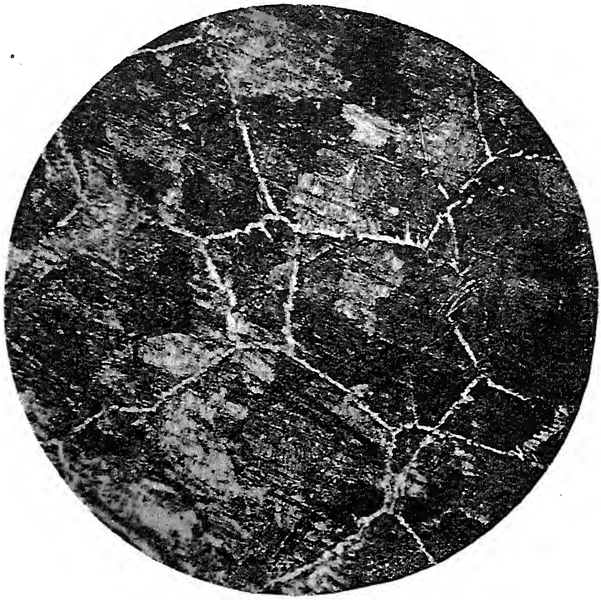

FIG. 1.—0.7 PER CENT. C STEEL HEATED TO 1100° C. FOR 5 MIN. AND FURNACE COOLED. × 100.

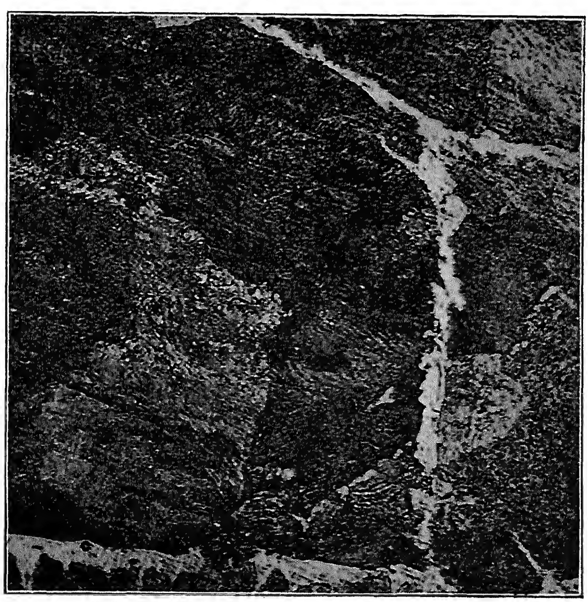

FIG. 2.—0.7 PER CENT. C STEEL HEATED TO 1100° C. FOR 5 MIN. AND FURNACE COOLED. × 400.

and 525 grains of pearlite per square millimeter. The ferrite consists of a large number of comparatively small grains arranged in such a manner as to form a rather complete network, surrounding the patches of pearlite. Each network surrounds, on the average, 14 grains of pearlite, *i.e.*, pearlite having the ferrite and cementite lamellæ running parallel in a given area. Some of these individual grains of pearlite can be seen in Fig. 1 and more distinctly in Fig. 2.

In the 0.7 per cent. C steel, just before the austenite changed to pearlite the ferrite network should have been substantially as it was after cooling to room temperature, *i.e.*, substantially the same as shown in Figs. 1 and 2; each of these ferrite network areas would have surrounded one grain of austenite; *but this one grain of austenite did not change into one grain of pearlite.* Calculated on the basis of areas, each grain of austenite changed into 14 grains of pearlite; or calculated on the basis of volumes, the result is 52, which represents the average number of grains of pearlite formed during the transformation of each austenite grain. In a similar manner it is found that each austenite grain has changed into 19 grains of ferrite when calculated on the basis of volumes.

The above structural analysis shows that there has been a decided refining of the grain size during the transformation of the coarse-grained austenite into ferrite and pearlite. What would happen if this steel were again heated above $Ac_3$? Since each austenite grain during the $Ar$ transformations broke up into many ferrite and pearlite grains, why would not these ferrite and pearlite grains break up into several austenite grains on heating above $Ac_3$? Each pearlite grain would change at $Ac_1$ into at least one or probably more than one austenite grain. Those austenite grains in contact with the ferrite at the network structure would dissolve the ferrite as the temperature increased and diffusion of the ferrite among the other austenite grains would immediately set in. Whether or not the diffusion would be complete would depend upon the temperature and the length of exposure above $Ac_3$. At the same time that the ferrite is in the process of diffusion among the comparatively small austenite grains, the latter may coalesce with one another, forming larger austenite grains. The ultimate grain size of the austenite would depend chiefly upon (*a*) the temperature reached and (*b*) the length of exposure at the elevated temperatures.

Assuming that the time of exposure is not excessive and that the highest temperature has not been much above $Ac_3$, on again cooling through the $Ar$ transformations, comparatively small ferrite networks would be formed at the boundary lines of the comparatively small austenite grains. This would produce what we know as grain refining in the steel. If the heating and cooling were repeated several times, an equilibrium grain size would result which would not be changed appreciably with any additional number of heatings and coolings under the

same time and temperature conditions. This equilibrium grain size is determined by (*a*) the equilibrium grain size of the austenite under the given time and temperature conditions, and (*b*) the number of ferrite networks and the number of ferrite and pearlite grains into which that austenite would transform at the given rate of cooling

This is why cast steel has its grain better refined after two heatings and coolings than after one.

It is well known wi th such steels as 0.7 per cent C steel that quenching from above $Ac_3$ produces martensite within which the excess ferrite (which would normally form on slow cooling) is quite evenly distributed. In such a sample, it might well be stated that the number of ferrite grains produced by the transformation of one austenite grain is such a large number as to border on the infinite. With intermediate rates of cooling such as would form sorbite, the ferrite particles exist in the form of isolated globules and each globule might be called one grain. Even in the case of the transformation of 0.7 per cent. C austenite into sorbite, the number of ferrite grains formed from one austenite grain would be exceedingly large—in fact, so large as to be indeterminate by present experimental methods

The rate of cooling has less influence on the grain size as the carbon content decreases and reaches its minimum influence in pure iron. The influence of the rate of cooling on a 0.7 per cent. C steel is great because the ferrite particles at the time of their formation are obstructed from coalescing with one another either by the intervening austenite or by the intervening transformation products other than ferrite.

What has been said about the effect of the rate of cooling on the ferrite grain size in a 0.7 per cent. C steel holds equally well for the pearlite grain size. The ferrite particles obstruct the pearlite particles and prevent the formation of large pearlite grains at the time of their (the pearlite's) birth from the austenite.

While the ferrite network outlines the old austenite grains in Figs. 1 and 2, the actual austenite grain boundaries have been effaced. The network of ferrite is formed of ferrite sheets of appreciable thickness. The thickness can be ascertained accurately from Fig. 2. The old austenite grain boundaries would, in general, about bisect the sheets of ferrite which form the network. In other words, the ferrite of the network has been obtained partly from one grain and partly from adjacent grains. This fact has been demonstrated by heating steels of this nature to a high temperature and holding at the high temperature for a long time and then cooling slowly to room temperature. Slag and other impurities would be collected at the austenite grain boundaries and would prevent the ferrite from abutting grains of austenite from coalescing. The resultant structure was a network of ferrite the sheets of which were about bisected, the bisecting lines representing the true austenite grain boundaries.

*0.2 Per Cent. Carbon Steel*

A piece of 0.2 per cent. C steel was heated to a temperature of 1100° C. held 2 hr. and cooled in the furnace. A micrograph of this steel at 200 diameters is shown in Fig. 3 The grain size of this steel after the above heat treatment was as follows:

Pearlite 210 patches per square millimeter.

Ferrite 580 grains per square millimeter.

The pearlite is given in terms of "patches" in order to determine in an indirect manner the minimum austenite grain size prior to the cooling of

FIG. 3.—0.2 PER CENT. C STEEL HEATED TO 1100° C. FOR 2 HR. AND FURNACE COOLED. × 200.

the steel. In such a steel it is not possible to find with the microscope the old austenite grain boundaries. It is known, however, that with steels of this nature, for example 0.2 per cent. C cast steel, each austenite grain may change on cooling into one or more than one patch of pearlite.

In certain cast steels, it is not uncommon for each austenite grain to change into many patches of pearlite and into many grains of ferrite. The same is true of meteorites, in which the Widmanstätten structure is pronounced. There should not, however, be fewer patches of pearlite than there were grains of austenite.

Since it is found, by grain-size determinations, that there are 2.8 times as many grains of ferrite on a given area as there are patches of

pearlite, it is evident that there are more grains of ferrite present than there were grains of austenite. This would be true even though it be considered that each patch of pearlite represents one austenite grain. It is very probable that there were more patches of pearlite in this steel than grains of austenite. Why, for example, would there be 36 grains of austenite per square millimeter with a 0.7 per cent. C steel heated for 5 min. to 1100° C., and 210 grains of austenite per square millimeter in a 0.2 per cent. C steel heated to the same temperature but held 24 times as long? We should expect a much larger austenite grain size under the above temperature and time conditions in the 0.2 per cent. C steel than in a 0.7 per cent. C steel.

Table 1.—*Austenite Grain Size in Carbon Steels as Determined by the Network Structure*

| Per Cent. Carbon | Temperature in Degrees Centigrade | Exposure | | Austenite Grains per Square Millimeter | Authoity |
|---|---|---|---|---|---|
| | | Hours | Minutes | | |
| 0 40 | 1,100 | .. | 3 | 100.0 | Howe* |
| 0.40 | 1,200 | .. | 10 | 17.0 | Howe |
| 0.40 | 1,200 | 2 | ... | 2.5 | Howe |
| 0.70 | 1,100 | .. | 5 | 36.0 | Jeffries |
| 1 14 | 1,100 | .. | 10 | 20.0 | Howe* |
| 1.14 | 1,100 | 1 | ... | 6.0 | Howe |
| 1.14 | 1,100 | 6 | ... | 2.5 | Howe |
| 1.14 | 1,200 | .. | 10 | 4 5 | Howe |
| 1.14 | 1,200 | 1 | ... | 3.0 | Howe |
| 1.14 | 1,200 | 2 | ... | 2.0 | Howe |
| 1.14 | 1,300 | 1 | ... | 1.4 | Howe |

* Taken from Howe's Table III, *Proceedings of American Society for Testing Materials* (1911), **11,** 344.

The data in Table 1 show that the austenite grain size increases with (*a*) the temperature and (*b*) the length of exposure. By interpolation from the table, the austenite grain size of Fig. 3 should have been about 5 grains per square millimeter. But this sample after furnace cooling measured 580 grains of ferrite per square millimeter. This is a decided grain refining and indicates non-inheritance.

If there is the slightest doubt about the change of these large austenite grains into many grains of ferrite, it will be removed by referring to Prof. Howe's micrographs,[2] of 0.4 per cent. C steel. Even the history of the grain refining can be studied. In Fig. 19, for instance, the austenite grain size can be traced from the ferrite network, while in Fig 21 it is not easy to tell where the old austenite grain boundaries were, but it can

[2] *Proceedings of the American Society for Testing Materials* (1911), **11,** Plate 1, Figs. 18–24

be seen plainly that each austenite grain has changed into several patches of pearlite and several grains of ferrite.

It is, therefore, evident that each austenite grain in the 0.2 per cent. C steel is broken up during the *Ar* transformations into several grains of ferrite and possibly into several patches of pearlite. If the latter is not true in this particular sample, we know it to be true in very coarse-grained samples of 0.2 per cent. C steel, for example, in cast steel. It is, therefore, evident that the grain size of the ferrite and the pearlite in 0.2 per cent. C steel is not inherited from that of the austenite.

The same line of reasoning regarding the rate of cooling and the grain refining of 0.7 per cent. C steel holds for a 0.2 per cent. C steel.

What has been said concerning hypoeutectoid steels also holds for hypereutectoid steels. A good example of a hypereutectoid steel, showing the formation of several grains of pearlite from one grain of austenite, is found in Prof. Sauveur's[3] Fig. 151, page 130.

Regarding the subject of grain size refining in steel, I will quote from Edward's *The Physico-Chemical Properties of Steel* (1916), page 124:

"This refining of the structure is caused by the recrystallization, which occurs as the metal passes through the critical ranges of temperature. The most probable explanation of this operation is as follows: Starting with an overheated steel possessing a structure as shown in Fig. 118, on heating to just above the $Ac_1$ point, the dark pearlitic areas become transformed into the $\gamma$-iron solid solution. Each of those areas do not, however, possess a uniform orientation—*i.e.*, consist of one crystal—but of a very large number of minute crystals of differing orientation. This recrystallization on heating is in many ways similar to the formation of small crystals from a liquid as it passes through its freezing temperature. When the steel is further heated, the initial crystals of the solid solution which contain 0.9 per cent. of carbon become capable of holding more iron in solution. Hence the ferrite surrounding these crystals is progressively taken into solution as the temperature is raised. Redistribution of the ferrite and carbon proceeds in this way, by diffusion, until the temperature of the highest recalescence point is reached, when the carbon is uniformly distributed through the mass. Whilst the steel is then of the same constitution as it was during the time of overheating, owing to the recrystallization which occurred on heating through the $Ac_1$ point and upward, it possesses an infinitely finer crystalline structure. Hence, when the metal is again cooled these small crystals behave quite independently by depositing the ferrite they contain in relatively small masses, and finally leaving only small areas of pearlite."

In the same chapter of Edward's book, there can be found some good micrographs illustrating the grain refining of steel.

### *Armco Iron*

Some experiments were made with Armco iron—the purest variety of commercial iron—with the idea of finding the relation between the ferrite grain size and the rate of cooling past $Ar_3$. The Armco iron used

[3] A. Sauveur: *The Metallography and Heat Treatment of Iron and Steel,* 1916.

in these experiments was furnished by the American Rolling Mill Co. of Middletown, Ohio. It is so free from carbon that there is not the slightest indication of martensite in a sample quenched in water from a temperature of 1300° C. In other words, with all rates of cooling from above the transformation range, ferrite is practically the only constituent formed.

Four samples of ¼-in. bar Armco iron were heated in an electric furnace in an atmosphere of hydrogen to 1300° C. and held for 2 hr. Samples A and B were withdrawn from the furnace and quenched in water and samples C and D were allowed to cool in the furnace. Grain-size measurements were made on each piece, after which the samples were again heated to 1300° C. and held 2 hr. After the second heating, samples C and D were withdrawn and quenched in water and samples A and B were allowed to cool in the furnace. It took the furnace, in each case, 45 min. to cool from 1300° C. to a black heat. Five or six grain-size determinations were made on each sample and to indicate the degree of uniformity of grain size, all of the measurements will be given.

*Heated in Hydrogen for 2 Hr. at 1300° C. and Quenched in Water*

| Sample A Grains per Square Millimeter | Sample B Grains Per Square Millimeter |
|---|---|
| 49 2 | 71 |
| 62.0 | 51 |
| 50 0. | 55 |
| 56.0 | 64 |
| 36.0 | 50 |
| 50.0 average | 58 average |

| Sample C Grains Per Square Millimeter | Sample D Grains Per Square Millimeter |
|---|---|
| 74 | 54 |
| 68 | 61 |
| 78 | 45 |
| 44 | 30 |
| 60 | 45 |
| | 50 |
| 65 average | 47 average |

*Samples Heated in Hydrogen for 2 Hr. at 1300° C. and Cooled in the Furnace*

| Sample A Grains Per Square Millimeter | Sample B Grains Per Square Millimeter |
|---|---|
| 11 | 15 |
| 11 | 19 |
| 18 | 18 |
| 12 | 16 |
| 6 | 24 |
| | 14 |
| 12 average | 18 average |

| Sample C Grains Per Square Millimeter | Sample D Grains Per Square Millimeter |
|---|---|
| 28 | 23 |
| 20 | 30 |
| 28 | 35 |
| 21 | 32 |
| 20 | 25 |
| 23 average | 29 average |

Fig. 4 is a micrograph, at 100 diameters, of one of the samples of Armco iron after furnace cooling from 1300° C., and Fig. 5 is a micrograph at 100 diameters of the same sample quenched in water from 1300° C. The difference in grain size can be seen readily from these micrographs. Calculated on the basis of areas, the grain size of the furnace-cooled samples is on the average 2.68 times that of the quenched samples and when calculated on the basis of volumes, each grain in the furnace-cooled samples corresponds to 7.2 grains in the quenched samples.

These results show conclusively that the ferrite grain size in Armco iron is not inherited from that of the austenite. If this were true, the rate of cooling from 1300° C. would have little or no effect on the ferrite grain size. The magnitude of the difference in grain size, assuming inheritance, would be represented by the amount of grain growth of the austenite during furnace cooling from 1300° C. down to 900° and the grain growth of the ferrite grains between 900° C. and room temperature. With such large grains of ferrite, it is probable that there was no grain growth below the transformation range on cooling. A piece of this Armco iron was cold-worked and then heated to 850° C. for various lengths of time, with the results shown in Table 2.

TABLE 2

| Length of Exposure at 850° C., Hours | Grains per Square Millimeter |
|---|---|
| 1 | 318.0 |
| 10 | 70.0 |
| 168 | 2.5 |

FIG. 4.—ARMCO IRON HEATED TO 1300° C. FOR 2 HR. AND FURNACE COOLED. × 100.

FIG. 5.—ARMCO IRON HEATED TO 1300° C. FOR 2 HR. AND QUENCHED IN WATER. × 100.

The average number of grains per square millimeter in the slowly cooled Armco iron samples is 20. If the same material consisting originally of very small grains, such as would be generated on recrystallization after cold plastic deformation, has 70 grains per square millimeter after an exposure of 10 hr. at 850° C., it is not at all likely that these furnace-cooled Armco iron samples would permit of appreciable grain growth on cooling from 900° to 850° C., and most certainly no grain growth could occur in these coarse-grained samples on cooling from 850° to room temperature.

It is also probable that there was substantially no grain growth on cooling from 1300° to 900° C. The reason for the latter is, that after a 2-hr. sojourn at 1300° C., austenite grains of sufficient size would have formed to check coalescence at temperatures lower than 1300° C., especially when the short time periods at the temperatures below 1300° C. are considered.

It is evident that the ferrite grain size in Armco iron is not inherited from that of the austenite, because with a given austenite grain size the resulting ferrite grain size will depend upon the rate of cooling through the *Ar* transformations. If the ferrite grain size in Armco iron is not inherited from that of the austenite on rapid cooling, why should it be inherited with slow cooling? We know that the ferrite grain size in Fig. 5 has not been inherited from the austenite. Has the grain size in Fig. 4 been inherited from the austenite?

By referring to Table 1, it will be seen that a grain size similar to that in Fig. 4 can be obtained in a 0.4 per cent. C steel in less than 10 min. at 1200° C., or in a 1.14 per cent. C steel in 10 min., at 1100° C. The former steel had its austenite grain size increased far beyond (2.5 : 20) that in Fig. 4 after a 2-hr. exposure at 1200° C., while under the same conditions the 1.14 per cent. C steel had a similar grain size to the 0.4 per cent. C steel, namely 2.5 : 2 grains per square millimeter. Knowing the great effect increase of temperature has on grain growth, it does not seem reasonable to suppose that the sample shown in Fig. 4, after a 2-hr. exposure to 1300° C. (100° higher than the above examples) would have so small a grain size, namely, 20 per square millimeter. The only conclusion is that the ferrite grain size in Fig. 4 is smaller than the austenite grains from which the ferrite was born—that is non-inheritance, and in this instance a tendency to refine the structure during cooling.

If the ferrite grain size in Fig. 4 is inherited from that of the austenite, what is the exact rate of cooling necessary in order to cause grain-size inheritance? Would not the ferrite grain size vary with every rate of cooling through the transformation range? It seems to me that it would.

If the old austenite grain boundaries are effaced during the *Ar* transformations in 0.7 per cent. C steel, and in a 0.2 per cent. C steel, why should they not be effaced in 0.02 per cent. C steel or even in pure iron?

Is it not a habit during the transformation of austenite, for the old austenite grain bound ariesthemselves to be effaced? If the ferrite collecting at the austenite grain boundaries in 0.7 per cent. C steel and in 0.2 per cent. C steel from two adjacent austenite grains coalesces, thus effacing the austenite grain boundary, why will not the same laws hold in 0.02 per cent. C steel or even in pure iron? I believe they would.

It should not be expected from a theoretical standpoint that a large grain of gamma iron should effect its transformation at the $Ar_3$ point into non-gamma iron from one nucleus only. It should normally be expected that such a transformation would begin from several nuclei; in general, the faster the rate of cooling the more nuclei. We know this to be true in steels in which the austenite is changed into sorbite by rather rapid cooling. Why should not the same laws obtain for pure iron? In the case of steel from which sorbite is formed by rapid cooling, the growth of the ferrite globules or grains is prevented by the obstruction of the other transformation products. In the case of pure iron, however, there is practically no mechanical obstruction to grain growth—in fact, the ferrite nuclei could absorb the adjacent transformed austenite, atom by atom as it changed into ferrite. For this reason, the purer the iron, the more rapidly will large grains of ferrite form during the transformation of the austenite. Since one grain of austenite must effect its transformation from at least one nucleus, it follows, axiomatically, that large austenite grains favor the formation of few nuclei and small austenite grains favor the formation of many nuclei. When but a few nuclei are formed, each grain resulting therefrom is probably of sufficient size to resist coalescence. Where many nuclei form, however, the resulting grains, being small, would have a tendency to coalesce until an approximate equilibrium grain size resulted. In any case, the nuclei which had formed at the austenite grain boundaries would feed upon the ferrite resulting from the $Ar_3$ transformation of abutting grains of austenite. If this be true, pure iron when changed from the gamma to the non-gamma condition does not retain, in the resultant ferrite structure, the old austenite boundary lines.

The hypothesis of the non-inheritance of the ferrite grain size from that of the mother austenite explains the known results with iron and steel. These facts are typified in Stead's paper on The Crystalline Structure of Iron and Steel[4] as follows:

"When pure iron, made coarsely granular by long heating at a dull red heat, is heated between 750° and 850° C. as a rule the structure is not altered to any material extent but as soon as the temperature is raised to 900° C. the granules again become small and heating to 1200° C. does not apparently produce any difference in their dimensions."

It is not reasonable to suppose that the austenite grain size would

[4] J. E. Stead: *Journal of the Iron and Steel Institute* (No. 1, 1898), **53**, 145–186.

be the same at 1200° C. as at 900° C. The non-inheritance hypothesis is the only one that explains these facts.

*Electrolytic Iron*

The discussion of electrolytic iron will be based largely on the work of Stead and Carpenter,[5] Ruder,[6] and results given in Group 2 of Prof. Howe's Table 1.[7]

Stead and Carpenter found the following regarding electro-deposited iron.

1. If heated to a temperature between 930° and 1000° C. for a moderate length of time and cooled in the furnace, a very coarse ferrite grain size usually resulted.

2. When samples heated as in (1) were quenched in water, *i.e.*, cooled rapidly through the $Ar_3$ transformation, a small ferrite grain size resulted.

3. If the time of exposure at 950° C. was increased to 20 hr. and the iron then furnace-cooled there was "a decided fining of the structure."

It is axiomatic that the grain size of the austenite (non-gamma iron) after 20 hr. exposure at 950° C. was greater than the grain size of the same material after 1 hr. exposure at the same temperature. With the same rate of cooling, however, the former (the larger grains of gamma iron) produced small ferrite grains while the latter (small grains of gamma iron) produced large grains of ferrite (non-gamma iron). It is thus seen that in electrolytic iron, at least under certain conditions, the ferrite grain size is not inherited from that of the austenite (gamma iron). It is also true that under these same conditions small grains of gamma iron (carbon-free austenite) tend to produce, with comparatively slow cooling, large grains of ferrite and that the comparatively large grains of gamma iron tend to produce, with the same rate of cooling, small grains of ferrite. Ruder obtained the same results as Stead and Carpenter.

The formation of large ferrite grains from small grains of gamma iron is what Prof. Howe calls "natal coarsening." It is evident that such coarsening must be natal, *i.e.*, born during the transformation from gamma to non-gamma iron. It is not necessary, however, that each austenite grain change into one ferrite grain (inheritance) to cause this natal coarsening. In fact, we have ideal conditions for coarsening according to the germinative temperature laws,[8] because each of the

[5] J. E. Stead and H. C. H. Carpenter: Crystallizing Properties of Electro-Deposited Iron. *Journal of the Iron and Steel Institute* (No. 2, 1913), **88**, 119.

[6] *Trans.* (1916), **56**, 589, 597.

[7] This volume, p. 489.

[8] H. M. Howe: On Grain Growth. *Trans* (1916), **56**, 582.

Z. Jeffries: Discussion on Recrystallization of Cold-Worked Alpha Brass on Annealing. *Trans.* (1916), **54**, 658.

Z. Jeffries: Grain Growth Phenomena in Metals. *Trans.* (1916), **56**, 571.

already small grains of gamma iron would tend to break up on cooling through the transformation range into at least one, or more likely several, grains of ferrite and these ferrite grains being (*a*) small, (*b*) having no mechanical obstruction between them and (*c*) formed at a time when a temperature gradient transversed the piece of iron, would form ideal conditions for grain coarsening by germination. All of these conditions would help to produce coarsening from small austenite or gamma-iron grains. When the gamma-iron grains are larger, however, they would tend to break up into less ferrite grains at the transformation temperature and hence ferrite coalescence would be much impeded because of the larger size of the individual ferrite grains. As the austenite or gamma-iron grains become larger and larger, the larger will be the ferrite grains which are formed by the partitioning of the austenite grains.

Stead and Carpenter have found that the rate of formation of the large ferrite grains under the natal coarsening conditions is extremely rapid. In fact, the gamma iron has to be fairly quenched in water to prevent the formation of the large ferrite grains. If the ferrite grain growth is as rapid as Stead and Carpenter have shown from transformed fine-grained austenite necessitating the coalescence of many individual grains of ferrite differently oriented or at least the agglomerating into one grain of ferrite many small grains of austenite, it seems that the rate of growth of a ferrite grain where the ferrite is supplied directly, as transformed from one or two austenite grains (implying substantially uniform orientation) would be even more rapid.

All of the results given in Prof. Howe's Table 1 can be explained with the hypothesis of non-inheritance of the ferrite grain size from that of austenite when considered in the light of the above discussion. For example, with very pure iron and with very small grains of austenite, germinative grain-growth laws may produce large grains of ferrite at the $Ar_3$ transformation. This effect may be slight or marked, the former escaping notice and the latter producing natal coarsening. As the gamma grain size becomes larger, the ferrite grains produced at $Ar_3$, although not inherited from the austenite grain size absolutely, are larger than would be produced from smaller austenite grains. The larger the austenite grain size, the larger will be the ferrite grain size with a given rate of cooling, but the latter are not inherited. In fact, it seems that the transformation of large grains of austenite into ferrite nearly always, under ordinary cooling conditions such as air- or furnace-cooling, produces more ferrite grains than there were grains of austenite. It may happen that the grain size of the ferrite is substantially the same as the grain size of the austenite from which it was born at $Ar_3$, but if so, the actual grain boundaries of the ferrite will not correspond with the grain boundaries of the mother austenite.

Regarding the question of grain-size inheritance, I will quote from Prof. Howe's discussion of Stead and Carpenter's paper.[9]

"Certainly the network of medium carbon hypoeutectoid and of hypereutectoid steel does inherit the coarseness of the austenite grains whence it springs. But it does not at all follow that the ferrite grains of ultra low-carbon steel should do likewise, for though the ferrite which composes them has been ejected by the shrinking mother austenite quite as the network ferrite is, and though that ferrite network of medium-carbon steel must needs reproduce the outline of the austenite grains which evolve it by expelling it to their own periphery, and though that quantity of the ferrite of dead-low carbon steel which is expelled thus from a given austenite grain may indeed have some form of solidarity, some common colonial trait, yet it need not form one ferrite grain: it need not inherit the austenite grain size. Instead, that ferrite, as it is progressively ejected from the shrinking austenite, may form a series of small grains, without relation to the grain size of the mother austenite.

"And when, because of their purity, the austenite grains do not become oversaturated with any impurity, oxide, sulphide, nitride, or otherwise, before cooling as far as $Ar_3$, and hence do not expel it and so encase themselves in a cement which isolates one from another, the orienting power of the nascent ferrite crystals may be able to cross those lines which had been austenite boundaries, so that the new crystals may be much greater than the austenite grains whence they spring."

The following data are taken from Stead's paper (reference 4).

TABLE 3.—*Showing the Effect on the Structure of Nearly Pure Iron (0.013 Per Cent. C) by Heating to Different Temperatures*

| Treatment of Iron 0.013 Per Cent. C | Manner of Cooling | Diameter of Average Grains in Millimeters | Grains per Square Millimeter |
|---|---|---|---|
| Rapidly heated to 1300° C...... | Cooled out of fire | 0.152 | 44 |
| Rapidly heated to 1000° C..... | Cooled out of fire | 0.152 | 44 |
| Rapidly heated to 950° C....... | Cooled out of fire | 0.152 | 44 |
| Rapidly heated to 800° C.. ... | Cooled out of fire | 0.228 | 19 |
| Heated 4 hr. to 1000° C. ........ | Cooled out of fire | 0.110 | 83 |
| Heated 4 hr. to 950° C. ... ... | Cooled out of fire | 0.125 | 64 |
| Heated 4 hr. to 800° C. ....... | Cooled out of fire | 0.223 | 20 |
| Before treatment............... | ............... | 0.228 | 19 |

Here again, as shown in Table 3, a nearly pure iron with entirely different origin (this iron was decarbonized in a furnace) from the electro-deposited iron, shows its smallest ferrite grain size from the largest austenite grains.

It is a very significant fact that observations from three varieties of nearly pure iron, namely, Stead and Carpenter's electro-deposited iron, which has been melted, cast, worked and reheated, Ruder's compressed slug of electrolytic iron powder which had never been heated as high as its melting point, and Stead's decarbonized iron, which was not of electro-

[9] *Journal of the Iron and Steel Institute* (No. 2, 1913), **88, 154.**

lytic origin, show the tendency, either slight or marked, to form larger ferrite grains from the transformation of fine-grained austenite, than spring from somewhat larger grains of austenite. It is probably true that with a gradually increasing size of the austenite grains a point will be reached at which the transformed ferrite grains will be larger, as the austenite grains become larger, for example, with austenite grains formed by long heating above 1000° C.

If the grain-size observations on electro-deposited iron are linked with those of compressed iron powder and decarbonized iron of non-electrolytic origin, the phenomena can be interpreted in accordance with the general laws of grain growth and grain formation from nuclei (which may form during the transformations in iron and steel), thus avoiding the embarrassing position that electro-deposited iron must act in a different way from iron of other origin, because it retains, even after melting, certain characteristics that were denied iron not of electrolytic origin. Probably the chief inherent characteristic of the electrolytic iron is its purity.

In conclusion, I wish to thank Messrs. J. S. Vanik and E. G. Hollman, for help rendered on the effect of time on the grain size of Armco iron below $Ar_3$, and W. P. Sykes, for assisting with the remainder of the experimental work recorded in this paper.

## Discussion

Zay Jeffries (written discussion*).—I have read with much interest Mr. Ruder's discussion of Professor Howe's paper, "The Supposed Reversal of Inheritance of Ferrite Grain Size from that of Austenite." I am yet of the opinion that my first explanation[1] of Mr. Ruder's samples is substantially correct. In that explanation I did not intend, however, to convey the impression that "reversed inheritance" was the general rule nor even of common occurrence but rather that "non-inheritance" was the general rule.

Mr. Ruder suggests that a "perfectly formed" grain is an indication of inheritance. The grains in my Fig. 5[2] are perfectly formed and in fact more nearly equiaxed than those in Mr. Ruder's Fig. C,[3] yet the grains in the former are known not to be inherited but to have been formed by the partitioning of larger austenite grains. On close examination of Mr. Ruder's Fig. C the very shapes of the grain suggest partitioning. Again, the grain size (25 to 30 grains per square millimeter) is at least 10 or more likely 20 times smaller than might be expected of the

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* Received Jan. 17, 1918.

[1] *Trans.* (1917), **56**, 594, 597.

[2] *Ibid.*, 580.

[3] *Ibid.*, 591.

austenite from which these grains were formed under the given time and temperature conditions.

Mr. Ruder's newer results on electrolytic and Armco iron are most interesting and instructive. His results fit in nicely with my conclusions 5, 6 and 10. His photographs, however, indicate that the thickness of the pieces of sheet metal are such that abnormal grain growth is promoted. He indicates that equilibrium grain size after heating to 950° C., with furnace cooling, is about the same as his Fig. 1, above. I estimate this to be approximately one-half grain per square millimeter, or 40 times as large as I have obtained with the same material (Armco iron) after heating ¼-in. bars to 1300° C. for 2 hr. and cooling in the furnace. The effect of thickness of sample on grain size is one which needs more attention.

Mr. Ruder's contention that compressed metal powders may have different characteristics from fused metals is entirely correct. The fact, however, that the consideration of compressed powders may complicate the general subject of grain-size inheritance after polymorphic changes, does not justify their exclusion from such a discussion. The compressed powders may not introduce nearly so great complications as the variation in the thickness of the samples.

W. E. Ruder, Schenectady, N. Y. (written discussion*).—To many members of the Institute, papers like this one by Prof. Jeffries, and that on "Reversal of Inheritance" by Prof. Howe, may seem highly academic. Aside from their direct application to all heat-treating processes, to my mind, the establishment of a physical fact and the correlation of facts into a definite law are in themselves ample justification for immense effort. A law, once established, cannot be ignored by those who carry the work along to practical ends. In studying both papers, it seems to me that each author is more concerned with defending a theory than with explaining the individual results as published by various investigators.

Take, for example, my sample of pressed electrolytic iron, frequently referred to by both. Prof. Howe, defending inheritance, says that the grains on the second (1300° C.) heating inherited their size from the mother austenite. Prof. Jeffries, defending reversed inheritance, elaborately explains it by saying that large gamma grains (1300° C.) resist coalescence much more than small gamma grains (1000° C.). As a matter of fact, I believe that so-called inheritance had nothing to do with the case.

As I admitted in a previous discussion,[4] the selection of pressed powders was unfortunate, as it unquestionably introduced unnecessary variables.

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* Received Feb. 18, 1918.

[4] This volume, p. 490.

Consequently, I took up the study of electrolytic iron which had been fused in vacuum and rolled into sheet 0.020 and 0.010 in. thick. I found that both types of structure were natal in the sense that they were both formed from alpha iron. The difference, as stated in my previous discussion, was this, that the large (1000° C.) grain was formed on heating and the smaller ones on cooling. I say they are both natal because, if quenched from above the $Ar_3$, the grain is extremely fine, so that the coarsening (as in Stead and Carpenter's experiments) occurred in the alpha iron exclusively and did not exist while the specimen was in the gamma phase.

What then determines the size of the grains? Exactly the same laws that our previous experiments (Mathewson and Phillips, on brass; Jeffries and Howe, on laws of grain growth; Ruder, on grain growth in silicon steels) have established. To be specific, the large grain was formed by the gradient (strain or temperature) which started growth at the point of maximum strain and as the temperature rose, successive areas became germinative. This grain was therefore formed, in all probability, long before $Ac_3$ was reached; so Jeffries' explanation of the arrangement of fine gamma grains into coarse alpha grains does not apply. I have tried this frequently and know that enormous grains (¼ by 1 in.) can be formed far below $Ac_3$ in the purest iron (as I have previously shown in silicon steel) by strain gradient, or even larger by moving a uniformly strained strip through a hot furnace at a rate lower than that of grain growth for the temperature used.

Now let us consider the upper end of the range. In these sheets the alpha grains are formed not more than 3 sec. after passing $Ar_3$ (my experiments here confirm those of Stead and Carpenter). If water-quenched from a high temperature (1300° C.) their rate of cooling is too rapid for growth and they are very fine (0.0076 sq. mm.). In other words, they are carried through their germinative range so rapidly that the whole area becomes germinative at the same time and the grains starting from a multitude of points mutually limit each other, just as they do if carried upward through their germinative range suddenly as by introduction into a hot furnace (below $Ac_3$).

If the piece is irregularly cooled (temperature gradient) or under strain during the passage from gamma to alpha, then columnar crystals are formed, because, near the point of most rapid cooling, an area attains its germinative temperature while those outside it are yet inactive; hence these germinative grains grow at their expense and at an extremely rapid rate, probably at least ½ to 1 in. per second, depending upon the gradient. Water-quenched samples have an average grain size of 0.0076 sq. mm., while 0.19 sq. mm. is the average grain size of air-cooled samples (0.010 in. thick).

Now consider the region outside the range of the gradient. Here we

have an area without strain or temperature differences except slight differences in cooling rates. What happens here? We have the same uniform condition as existed in the quickly cooled area, but longer time in the germinative range has allowed for grain growth to considerable coarseness.

It will be unnecessary to show photographs of these different phases, as Stead and Carpenter's photographs,[5] particularly No. 3, plate II, are almost identical with my samples. In fact, I am in entire agreement with all of Stead and Carpenter's results, except their statement that these facts apply only to electrolytic iron, for I have obtained similar results with hydrogen-purified ingot iron. I also take exception to their conclusion (4) "that once coarse crystals are formed they cannot be destroyed except either by cold mechanical work, or by heating above $Ac_3$, followed by quenching."[6] My experiments show that the crystals are changed every time they cool from $Ac_3$, and that their size depends entirely upon the rate of cooling through the $Ac_3$–$Ar_3$ range; also, that the columnar grains formed by a strain gradient are destroyed by merely heating through $Ac_3$ without any quenching, unless, of course, the same conditions of gradient are applied again on the second cooling.

These general laws hold regardless of thickness of the strip. The condition of time, strain gradient, and rate of growth are different, however. For example, 0.020-in. strips quenched from 1200° C. have a grain size about six times those of strips 0.010 in. thick, and show columnar grains up to ¼ in. long, while the thinner strip shows no sign of columnar structure under these conditions. My only explanation for this is that the thicker strip cools more slowly, even on quenching. No marked difference was noted on quenching from 1300° C. or from 950° C. It will be interesting to investigate this to greater thicknesses, to see if the only difference is the effect upon the cooling rate. In fact, I have not yet made enough experiments to offer any but tentative remarks about the effect of thickness.

With these facts before us, there seems to me absolutely no reason to believe that there is any connection whatever between the gamma grain size and the alpha grain size. On heating past $Ac_3$, or on cooling past $Ar_3$, we have in each case a clean slate upon which to write, and the message written there is entirely dependent upon the conditions existing at the instant (almost) of change. It is like starting with a new sample altogether. We have always assumed that the grain refines on passing $Ac_3$ on account of a change of crystal form. Why have we not logically admitted that if grain is refined when changing to gamma iron, it must

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[5] J. E. Stead and H. C. H. Carpenter: The Crystallizing Properties of Electro-Deposited Iron. *Journal of the Iron and Steel Institute* (No. II, 1913), **88**, 119–170.

[6] *Op. cit.*, 144.

also go through the same process to regain its cubical alpha form? It seems to me that the whole idea of inheritance must be a myth inherited from disproved ideas about grain growth, and inconsistent with our newer ideas.

So much for pure irons, which are the only ones of which I can speak from experience. I find I have arrived at the same conclusion as the present author, even though I have taken a somewhat different path.

Prof. Jeffries seems to take the ground in all of his arguments that the change from gamma to alpha (or *vice versa*) crystals is a direct one and influenced to some extent by the previous state. In this I differ from him in that I maintain that the change always takes place through a period of ultimate refinement which, for the instant, makes the entire piece like one from which all previous history is wiped out.

Of course, in carbon steels such as shown in the author's Fig. 1 and 2,[7] we have a non-homogeneous mass, and the transformation change is so rapid that actual diffusion of the segregated ferrite could not possibly take place; and the unit in this case must be considered the austenite area which the author has also shown to be unrelated to the *pearlite formed.* For the carbon steels, I feel that the author has presented his case extremely well and the conclusions bear out the general proposition of non-inheritance.

As for the silicon steels, from which I have been able to produce, at will, crystals as large as 2 by 8 in., these must be omitted altogether from any discussion of inheritance because they have no $Ac_3$ point. No heat treatment, short of actual fusion, will refine these grains; whence I assume that the gamma state is above the melting point of the steel. The laws governing the growth of grains on heating, however, are exactly the same as in the case of pure irons, namely: (1) Each degree of strain has a corresponding germinative temperature, or range, as I prefer to call it. (2) This germinative range decreases as the strain increases. (3) That the presence of foreign materials or carbon in its different forms or compounds, acts as a retardant to grain growth.

Zay Jeffries,* Cleveland, Ohio (written discussion†).—Mr. Ruder has given us some new and important data on grain growth in iron and low-carbon steel. While I agree with Mr. Ruder in some of his conclusions I disagree with him in others.

The contention that his large (1000° C.) grain formed on heating is inconsistent not only with his general conclusions regarding non-inheritance but also with the results of every other investigator in this field. The fact that large grains can be produced in iron below $A_3$,

[7] See p. 671.

* Director of Research, Aluminum Castings Co.

† Received Feb. 25, 1918.

with proper strain gradient, has nothing to do with the subject of inheritance on passing an allotropic change point except as the grain size of one allotrope influences that of another. Cooling strains, and those caused by the volume changes during the transformations, might play a rôle in grain growth, but if I understand Mr. Ruder correctly he did not have this type of strain in mind. In order to have been formed in the non-gamma iron on heating, the large grain in question must have changed into gamma iron at $Ac_3$ without change in size or shape; it must have maintained that size and shape during a 3-hr. sojourn at 1000° C., notwithstanding the fact that it would have been in contact with smaller grains, and finally conditions of absolute inheritance must have existed during the $Ar_3$ transformation on cooling. This is not only contrary to the findings of Stead, which have been confirmed many times, but also contrary to Ruder's own reported experimental results. If this were true the same conditions should have existed, or at least the grains should not have become smaller with the 1300° C. heating of the same sample after the large grain had been formed.

Mr. Ruder seems to believe in absolute non-inheritance as opposed to either absolute inheritance or relative inheritance. Prof. Howe believes in either absolute or relative inheritance. The writer believes in non-inheritance as opposed to absolute inheritance, but believes also that the grain size of one allotrope influences that of another into which it transforms. It seems to me that this law is always operative according to conclusions Nos. 5 and 6, but the following factors may either destroy the results of relative inheritance or even produce what Prof. Howe calls "reversed inheritance."

According to conclusion No. 11, there will always be a tendency to produce more grains in a new allotrope than existed in the old one just previous to a transformation which involves a change in crystal form, which normally occurs in a temperature region in which grain growth in both allotropes is rapid. A grain of austenite or gamma iron transforms into other allotropes most readily from the boundary. This is shown in steels because the excess ferrite and cementite represent the first stage of transformation and these form first at the austenite grain boundaries. The cleavage planes of the austenite offer the next best place for nuclei of transformation to form. We might compare a grain of austenite to a great number of bricks piled in the same orientation, and ferrite to the same bricks piled regularly but in a different arrangement. If we wish to change the orientation of one of the interior bricks from one arrangement to the other, would it not be the easiest way to begin at the outside of the pile and first change the arrangement of all bricks up to the particular one? If the rate of transformation from the grain boundary inward is not sufficient to keep pace with the forces tending to produce the transformation in the interior of the grain, then, and only then, will interior

nuclei form. These nuclei would be similar, in pure iron, to the excess ferrite and cementite in the cleavage planes in the austenite of carbon steel. We should not expect interior nuclei to form unless either the grains of austenite are large or the cooling through the *Ar* points is rapid, or both.

We should therefore expect fine-grained gamma iron to transform into more grains of non-gamma iron at $Ar_3$ than would coarse-grained gamma iron, because the latter has much less grain boundary surface. We should also expect considerable refining, that is, more grains of non-gamma iron than there were of gamma. But the non-gamma iron at and well below the $A_3$ point, is in a region of rapid grain growth; hence, if its grain size immediately after its transformation is less than the equilibrium grain size for the particular material at the given rate of cooling, grain growth will take place during cooling.

So far as the grain size relations between the gamma and non-gamma iron are concerned, we may divide the former into two ranges.

Range 1. The number of nuclei in a given volume from which the gamma iron transforms is greater than the number of non-gamma iron grains in the same volume stable under the given cooling conditions. Grain growth in the non-gamma iron will then take place.

Range 2. The number of nuclei in a given volume from which the gamma iron transforms is equal to, or less than, the number of non-gamma grains in the same volume stable under the given cooling conditions. In this case grain growth in the non-gamma iron will not occur.

It is obvious that a sample might be in Range 2 with one rate of cooling and in Range 1 with another.

While there would be a tendency toward equal grain size in the non-gamma iron produced from gamma iron in Range 1, after cooling at a certain rate through and well below $Ar_3$, a temperature gradient would be more apt to produce exaggerated grain growth from the smaller grains than from the larger. During normal furnace cooling, probably grain growth of the non-gamma iron ceases when a temperature a few degrees below $Ar_3$ is reached.

These ideas accord with the general proposition that iron can be heated to temperatures from 900° to about 1100° C., and, when cooled at a given rate, there is no marked difference in grain size; but when cooled at the same rate from temperatures well above 1100° C., the grain size of the non-gamma iron increases as that of the gamma iron increases. The range between 900° and about 1100° C. would be Range 1 under these conditions.

The fact that Ruder noted no marked difference in grain size in 0.020-in. strips quenched from 1300° C. and 950° C. is no indication that the same samples, furnace-cooled, would have had approximately the same grain size. Quenching might put these pieces both in Range 1, while

furnace or other slow cooling might put the 950° C. sample in Range 1 and the 1300° C. sample in Range 2. Such experiments as I have conducted along this line indicate that, in general, the grain size of non-gamma iron produced by quenching gamma iron does vary decidedly with the grain size of the latter—the larger the grains in the gamma iron the larger will be the non-gamma iron grains.

It frequently happens than the grain size in non-gamma iron is larger than that in the gamma iron from which it was produced. For example, Humfrey[8] writes:

> It is necessary to follow the changes which go on in the crystal structure of iron when it is heated and cooled through the change points and concerning this we can deduce considerable information from the surface patterns which are produced on a polished sample during heating and cooling . . . It was found on examining such a sample that if it had been heated above the $Ar_3$ point two distinct networks were visible, one of which represented the gamma crystals of the iron and the other the boundaries of the alpha crystals which remained when cold. The two networks crossed each other in a quite irregular manner and generally the alpha network represented considerably larger individual crystals than the gamma. (Humfrey worked largely below 1000° C.). If the sample had been heated below the $Ar_3$ point only one network representing the final alpha crystals was visible. These facts show that the iron on passing through the $Ar_3$ change point undergoes a complete recrystallization, the boundaries of the gamma crystals above $Ar_3$ being entirely distinct from those of the alpha after this point has been passed.

Humfrey's results would indicate a greater resistance to grain growth in gamma than in non-gamma iron. He worked with electrolytic iron, transformer sheet steel, wrought iron, and "ferrum reductum" produced by reduction of iron oxide with hydrogen. When we compare Humfrey's results with those of Stead, Howe, Ruder, and others, it is seen that iron produced by practically every method known responds to the general propositions of change in grain size during the transformations as set forth in my paper and in the present discussion.

The fact that gamma iron has greater resistance to grain growth from 900° to 1000° C. than non-gamma immediately below $A_3$, is in accord with Prof. Howe's contention that inheritance is more pronounced on cooling than on heating The difference is not due to any change in inheritance laws but to the different resistances to grain growth of the gamma and non-gamma iron. A difference in the velocity of transformation may also contribute. I would consider the velocity of transformation equal to or less than the velocity of growth of non-gamma iron within a transforming grain of gamma iron which had not begun but which was on the verge of transforming from interior nuclei. This velocity must be greater than that of the growth of one non-gamma grain feeding upon smaller ones. If this is true, a smaller grain size will result from quenching fine-grained than coarse-grained gamma iron.

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[8] *Iron and Steel Institute, Carnegie Scholarship Memoirs* (1912), **4**, 80–107.

Samuel T. Hoyt,* Minneapolis, Minn. (written discussion‡).—The subject of grain inheritance in iron and steel is apparently one which can be considered from the point of view of the two laws governing the rate of formation of crystal nuclei and the linear velocity of crystallization, which have been investigated by Tammann and collaborators, and of the additional law of coalescence. According to the work of Tammann, both the nuclei number, which is taken to be the number of crystal nuclei to form per unit volume per unit time, and the linear velocity of crystallization vary with the temperature or degree of supercooling. Therefore, according to the rate of cooling, we may have different absolute and relative values of these two factors governing the formation of the new phase, or phases, as the case may be. If the maximum value for the linear velocity is obtained just under the transformation point, but above the temperature corresponding to the maximum nuclei number, slow cooling produces a larger grain size than rapid cooling. On the other hand, if the linear velocity does not attain appreciable magnitude until a temperature below that corresponding to the maximum nuclei number, we may have the reverse action, in which case rapid cooling produces coarser grain than slow cooling. The former of these two cases is the more common, as it is well known that rapid cooling ordinarily produces the finer grain size, although the rate of cooling itself has but the secondary effect of determining the actual values of the nuclei number and linear velocity which govern the crystallization of the new phase. It would then seem necessary to consider here the nuclei number and linear velocity as primary factors and the rate of cooling, chemical composition, chemical constitution, and physical condition, as secondary factors.

It is quite evident that pure iron recrystallizes at $A_3$; *i.e.*, crystal nuclei form in accordance with the momentary nuclei number and these crystal nuclei grow into crystallites of the new phase at a rate which is commensurate with the momentary velocity of crystallization, both of which may actually vary during the transformation. Only in case the grain size were smaller than the "equilibrium grain size" would coalescence play an important rôle in governing the grain size.‡ According to this, we would not expect an inheritance of grain size in pure iron, nor in the alloys of iron and carbon containing up to 0.3 per cent. C, the structure of which is largely determined by the structure of ferrite. In no case need we consider that one crystallite of $\gamma$ iron changes into one crystallite of $\alpha$ iron, or *vice versa;* not even grain-size inheritance would indicate this.

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* Assistant Professor of Metallography, University of Minnesota School of Mines.

‡ Received Apr. 1, 1918.

‡ I assume Professor Jeffries' definition of equilibrium grain size would be that grain size which is very readily assumed at any temperature but which is not very readily increased by further annealing.

In the case of the hyper- and hypo-eutectoid steels, grain-size inheritance is the natural consequence of the segregation of the ferrite and cementite and the lack of sufficient material for the formation of crystallites of normal size, as is true when the carbon is below 0.3 per cent. The formation of pearlite "colonies," while subject to the same laws, in common with the formation of eutectics and eutectoids in general, to the writer's knowledge, has not been worked out satisfactorily as yet. This is due principally to our lack of knowledge as to the mechanism of the process. However, it seems perfectly clear that the size of the individual colony will be different from the size of the austenite grain just as $\alpha$ iron will have a different grain size from $\gamma$ iron. On heating, the conditions are somewhat different, in that material from two different sources (ferrite and cementite) is used in the production of the new constituent, the solid solution. Here again our knowledge of the mechanism of the process is somewhat deficient, but it seems that the austenite to form is excessively fine grained (Hanemann). This indicates that an excessively large number of austenite crystallites (for it seems necessary to assume that this austenite is crystalline) form from the pearlite, which grow in size by coalescence in the case of eutectoid steel, or by coalescence and the dissolution of ferrite and cementite in the case of hypo- and hyper-eutectoid steels. In the latter case, growth by coalescence is enhanced by the rising temperature.

As an attempt to explain the subject matter and questions raised in the second paragraph on page 681, the following is advanced. The formation of ferrite particles in sorbite differs from the formation of ferrite crystallites in primary ferrite and in pearlite. When lamellar pearlite forms, there is evidently an appreciable velocity of crystallization, such that the nuclei readily grow to visible size. When the ferrite particles of sorbite form, the linear velocity is evidently very low, so that about all we have is the formation of a large number of ferrite particles a little larger than nuclei. Their growth, as for example in the granular pearlite, would then be due to coalescence. As to the relationship between the grains of austenite and the nuclei of the new phase, it should be remembered that the formation of nuclei is a property, not of the old-phase austenite, but of the new phase which is forming. With this in mind, it would be of interest to investigate the factors affecting the formation and growth of the transformation products.

W. E. Ruder (written discussion*).—I must admit Mr. Jeffries' contention, that my general conclusions would be inconsistent with my explanation of the structure of the pressed iron sample if I admitted that the pressed iron followed the laws of the perfectly orthodox and regular, fused and worked metals. This is, however, just what I do

* Received March 25, 1918.

not admit, and I have been trying my best to have this particular piece of evidence stricken from the court records. It seems, however, that I am to be estopped.

Judging from the appearance of the large grain which is entirely different from anything I have ever seen before, I conclude that it is not a real grain in the ordinary sense, but merely an agglomeration of particles themselves made up of crystals which, owing to their uniform strain, happen to be oriented in about the same way. This pseudo-grain, on account of its conglomerate nature, could not be expected to follow ordinary laws. I certainly cannot conceive its being in any way related to the finer structure obtained in reheating. I can easily conceive of such a mass of uncoalesced particles being held at 1000° C. for a long period without coalescing. The pressure to which they were subjected would be the most important factor in this case.

I would like to repeat, however, that this exceptional case is so different from the regular run of material that it is not worthy the attention it has received. I am not at all sorry that I called attention to it in the first instance, however, because it has been largely instrumental in drawing from Prof. Howe and Prof. Jeffries a very important discussion of the question of inheritance.

My belief in absolute non-inheritance is based upon the belief that iron changes crystalline form in passing into the gamma state. Satisfactory proof that this is the case is, I know, still lacking. However, there is evidence of that possibility and if the cubical crystal does change form, then Prof. Jeffries' simile of transferring a pile of bricks from one point to another does not hold absolutely. The bricks change shape before any transfer can take place, and it is the instant between the change of shape and that of rebuilding that, I contend, makes even relative inheritance improbable.

Mr. Jeffries has shown that the non-gamma grains may be larger or smaller than the gamma iron, depending upon given cooling conditions—moreover, he quotes from Humfrey to show that the gamma iron structure has no relation to the succeeding non-gamma structure. So far we agree, but I cannot see any reason for referring to nuclei as if they were some definite pre-existent thing predetermining, or in any way influencing, the grain size of the succeeding phase. To have grain growth, we must have a starting point, it is true. That starting point we call a nucleus. This nucleus cannot exist before its phase is achieved, and its existence is determined merely by the influences at work upon it at that instant.

## Time Effect in Tempering Steel

BY A. E. BELLIS,* S. B., SPRINGFIELD, MASS.

(New York Meeting, February, 1918)

The time effect in reheating certain steels below the critical range is very marked. The increased toughness, shock-resisting power, and machinability of steel subjected to a long, high drawing temperature has been thoroughly demonstrated and is of practical importance, particularly in the manufacture of "smokeless" rifle-barrel steel.

The time effect in tempering is most important when a maximum drawing effect is desired, in which case the highest physical properties as well as ease of machining are important considerations. These are the conditions in drilling a rifle barrel. A hole, 0.30 in. in diameter and 24 in. long for the Springfield rifle, and 30 in. long for the Russian military rifle, has to be drilled in the heat-treated material. This barrel-drilling operation is probably the most difficult one in rifle manufacture, and anyone who has been connected with rifle manufacture knows how seriously production is affected when "hard" or non-uniform steel for barrels is encountered. The importance of high physical properties (tensile strength, toughness and resilience) in a rifle barrel is obvious.

The physical properties of two different lots of barrel steel are given below. The first lot, No. 1, gave serious trouble in the drilling operation; the second gave no trouble at all.

| | No 1 | No 2 |
|---|---|---|
| Elastic limit, lb. per sq. in. | 117,450 | 116,800 |
| Tensile strength, lb. per sq. in. | 132,500 | 131,750 |
| Elongation, per cent. | 16.0 | 20.0 |
| Reduction of area, per cent. | 42.1 | 51.0 |
| Impact strength, ft.-lb. per sq. in. | 450.0 | 520.0 |

This difference in machining and physical properties was due entirely to the time of reheating. The first lot was given a reheating time of 30 min., the second lot remained at the reheating temperature 2 hr. Both

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lots had been oil-quenched from 1500° F. and reheated to 1180° F. The time required in the reheating or drawing for the work to come to temperature was not counted. This time was approximately 20 to 30 min. The steel used has the following composition: carbon, 0.54; sulphur, 0.050; manganese, 1.22; phosphorus, 0.065.

The physical properties when untreated are as follows:

| | |
|---|---|
| Elastic limit, lb. per sq. in. | 69,800 |
| Tensile strength, lb. per sq. in. | 128,700 |
| Elongation, per cent. | 15.0 |
| Contraction, per cent. | 35.2 |

Further experiments established the fact that at least a 2-hr. reheating is necessary in order to have satisfactory machinability. Steel which, after treatment, gave elongation of 20 per cent. or over gave no trouble in the shops. These results were confirmed by hundreds of physical tests and production reports on thousands of barrels.

It will be noted that there is a decrease in the elastic limit and tensile strength as the drawing time is increased, but that this is slight compared to the relatively great increase in ductility. The decrease of elastic limit of less than 1 per cent. is accompanied by an increase of 25 per cent. in ductility as measured by elongation; also by an increase of 13 per cent. in impact strength.

A similar test with different heat lots of steel of slightly different analysis indicates that further increase in machinability, ductility, and resilience can be obtained by making the drawing time still longer. Below are the results of this test (A) with drawing periods of 1, 2 and 3 hr.; together with the results of another test (B) made with drawing periods of ½ hr. and 12 hr. The latter test was made with the idea of producing the maximum time effect.

| | A | | | B | |
|---|---|---|---|---|---|
| | 1 Hr. | 2 Hr. | 3 Hr. | ½ Hr. | 12 Hr. |
| Elastic limit, lb. per sq. in. | 124,250 | 121,600 | 116,250 | 115,500 | 98,750 |
| Tensile strength, lb. per sq. in. | 137,000 | 135,600 | 125,900 | 135,400 | 116,500 |
| Elongation, per cent. | 17.0 | 17.5 | 19.0 | 17.5 | 22.0 |
| Reduction of area, per cent. | 42.2 | 45.4 | 47.6 | 52.7 | 57.2 |

In order to determine whether this effect was largely due to the high manganese content of the steels, an ordinary machining steel of the same carbon content was given a similar test. The steel analyzed: carbon, 0.53; sulphur, 0.036; manganese, 0.65; phosphorus, 0.049.

Four specimens were oil-quenched from 1500° F., and drawn at 1180° F. for periods of ½, 1, 2 and 4 hr. respectively. The results follow:

| | Reheating Time | | | |
|---|---|---|---|---|
| | ½ Hr | 1 Hr | 2 Hr | 4 Hr. |
| Elastic limit, lb. per sq. in. | 66,150 | 63,800 | 62,750 | 61,600 |
| Tensile strength, lb. per sq. in. | 101,250 | 98,250 | 97,250 | 96,600 |
| Elongation, per cent | 23 0 | 26 0 | 28.5 | 28 5 |
| Reduction of area, per cent. | 60 8 | 67 5 | 65.0 | 65 8 |

In all of these experiments, the standard test specimens were prepared from the bar stock *after* the heat treatment. The original diameter of the stock was 1⅓₂ in. The heat treatment was carried on in large production furnaces, temperatures being controlled with carefully standardized Wilson-Maeulen pyrometers. The reheating time was not counted for the first 20 or 30 min. necessary for the pieces to reach furnace temperature. A reheating time of ½ hr., therefore, means at least 50 min. in the furnace.

For quenching, a straight mineral oil was used of 0.890 specific gravity at 60° F., 400° F. flash point and 200 sec. Saybolt viscosity at 100° F.

Since our measurements of strength, ductility, and resilience are not absolute, and because of the inherent structural variations in steel, we cannot make exact conclusions from a few tests such as are given here. From daily contact, however, we acquire knowledge of these properties which we cannot easily record, but of which we are certain. I regard this knowledge from experience the best evidence that increasing the time of reheating increases the ductility, toughness, and machinability of the particular steels herein described, to a degree well illustrated in the above tests. The excellent physical properties that can be obtained by properly heat-treating a machinery steel containing 1 to 1½ per cent. manganese does not seem to have been generally recognized.

## Discussion

H. M. Boylston, Cambridge, Mass.—I would first like to ask what machine Capt. Bellis used to make his tests on impact strength. I have made some tests with the Charpy impact machine, the specimen being supported at both ends and nicked in the middle, and find that the results do not always agree with those obtained by the Izod machine in which the unnicked test piece is held at one end while the pendulum shears off the other end. I would also like to ask what effect the composition and heat treatment of the tool steel had on the drilling of his rifle barrels, as compared with the heat treatment of the barrel-steel itself?

I notice that the analysis of the composition of steel used shows sulphur 0.050 and phosphorus 0.065; I should like to ask how those high

limits of sulphur and phosphorus came to be allowed? I have some data which lead me to feel pretty certain that those high limits make for very unsafe steel in rifle barrels.

It has never been explained to the satisfaction of all of us, why the manganese is so high: 1.22 per cent. I believe several reasons have been stated; among them, that it is high in order to compensate for the high sulphur, and also to increase the elastic limit and improve other physical properties of steel.

Lastly, I wish to ask whether Capt. Bellis has tried any different methods of heat-treating his steel rifle barrels? Apparently these were all heat-treated before machining; has he had any experience with heat-treatment after machining, especially after drilling?

Those questions may seem somewhat outside of the range of this paper, but they are important, and if Captain Bellis can give us any information, it will be very helpful.

Dr. J. A. Mathews,* Syracuse, N. Y.—The same subject was discussed here last year, and the earlier papers by Dr. Carl Barus seem to indicate that nothing was gained by the length of time of tempering. (It should be borne in mind that his results were based upon electrical and magnetic measurements and not upon tensile tests.) That idea has been accepted as gospel by steel workers, until late years, and particularly in connection with alloy steels, on which there is no question that the effect of time of tempering is very marked.

We carried on an investigation of alloy spring steel a number of years ago, and the quality of the springs was very much better when we gave them a reasonable length of time in drawing. The old-fashioned method of flashing off the oil did not give very good products.

Carle R. Hayward,† Cambridge, Mass. (written discussion‡).—The figures given by Mr. Bellis are interesting and confirm previous information on the subject of tempering. In a paper by S. S. Raymond and myself, "Effect of Time in Reheating Hardened Steel,"[1] the following statement occurs: "At temperatures above 500° C., increasing the time of treatment causes a slight falling off in hardness and tensile strength with a corresponding increase in ductility."

The drawing temperature used by Mr. Bellis was 1180° F., or 638° C. His results at this temperature are comparable with those obtained at 600° C. in the paper cited above, but his ductility was slightly greater, as would be expected with the slightly higher drawing temperature.

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† Asst. Professor of Mining and Metallurgy, Massachusetts Institute of Technology.

‡ Received Feb. 27, 1918.

[1] *Trans.* (1916), **56**, 517.

# Some Structures in Steel Fusion Welds

BY S. W. MILLER,* M. E., ROCHESTER, N. Y.

(New York Meeting, February, 1918)

DURING the examination of welds made in steel by the oxy-acetylene and electric-arc processes, the writer has met with some unusual structures, which he has not encountered elsewhere. They seem to be quite closely associated with the welding methods used, and characteristic of them, and would seem to be worth noting, especially as the references to them in the literature are few, and their explanation not clear or complete. An introductory word or two about fusion welding may not be out of place.

The oxy-acetylene process is based on the great and very local heat generated by the burning of acetylene in the presence of commercially pure oxygen, the two gases being thoroughly mixed before combustion. The theoretical temperature is somewhat under 4000° C., but on account of various losses, the actual maximum temperature is probably about 3000° C.[1]

The electric-arc process, using a metal electrode, which is also the filling rod, generates a still higher and more local temperature. The actual temperature of the positive crater of the *carbon* arc is given as 3500° C.[2] No figures have been found for the temperature of the arc using a metal electrode, but it would probably not be very different from that of the carbon arc, and probably higher than that of the oxy-acetylene flame. The greater localization of the electric arc is due to the fact that it is not surrounded by a large envelope flame, as well as to its higher temperature; while the oxy-acetylene white cone, or welding flame, in which the carbon of the acetylene is burned to CO, is surrounded and preceded by a long envelope flame, itself of quite high temperature, in which the CO is burned to $CO_2$, and the hydrogen of the acetylene to $H_2O$. This flame spreads around and over the pieces being welded, heating them to a further distance from the weld than does the electric arc.

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* Proprietor, Rochester Welding Works.

[1] Letter from Professor Joseph W. Richards, Oct. 15, 1916.

[2] G. K. Burgess and H. L. Le Chatelier: *Measurement of High Temperatures*, 3d Ed., p. 454. N. Y., John Wiley & Sons, 1912.

These differences between the flames would naturally make some differences in the welds made by them. The greatest of these differences, the writer believes, is due to the more rapid heating and cooling of the electric weld, which causes much less coarsening of the grain of both the weld and the original material, and causes the coarsening to extend a smaller distance from the weld than does the oxy-acetylene process; so that the method of making a weld is readily determined, provided it be unannealed. The difference in grain size is seen in Fig. 1 and 2, and has been referred to many times.[3]

In low-carbon steel wire (carbon 0.10 per cent. or less) and in wrought iron, the cementite exists as small isolated particles at the grain boundaries, which appear as in Fig. 3 to 7 inclusive. Many times the particles are entirely dark, when etched in sodium picrate, but at other times they seem to have what might be called a type of pearlitic structure, which can be faintly observed in Fig. 5. The relief effect was obtained by throwing the light a trifle to one side in a horizontal plane, and is seen more readily than it can be photographed. Even the dark particles do not seem to be uniformly so at high powers, as Fig. 8 shows the lighter parts being reddish-brown. This piece was boiled 45 min. in sodium picrate. Also, there is frequently a variety of colors visible, ranging from dark red and brown to black, the usual color being brown. These colors are not visible at less than 400 diameters, as far as the writer has observed, less magnification showing simply darkening of the cementite, as in Fig. 3. The cause of these colors is not clear.

The general subject of cementite in low-carbon steel is mentioned because it is very important in welding; low-carbon material (not over 0.10 per cent. for oxy-acetylene, and 0.18 per cent. for electric) being usually added. The location and structure of the cementite has, it is believed, an important bearing on the strength of the weld, and it is, therefore, advisable to determine the conditions under which it exists, so that they may be bettered, if necessary, by heat treatment. The latter subject is a very broad one, and the writer has done nothing with it as yet. He has, however, noticed some peculiarities in the character of the cementite, which seem to merit attention, and which may be a starting point for heat treatment.

With both kinds of welding, there is an agitation of the melted metal. This is quite violent with the electric method, resulting in there being more or less oxidizing of the iron. In electric welding, there is also the condition that some of the metal passes from the electrode as a spray, materially increasing the oxidation. No fusion weld that the writer has examined has been free from considerable amounts of oxide, which

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[3] For example, see Law, Merritt and Digby: Some Studies of Welds. *Journal of the Iron and Steel Institute* (No. I, 1907) **95**, 115, and discussion.

can be seen in its usual form in many of the photographs; for example, Fig. 9, 10, 22, 23, 27, 34, 37, and 42. It seems probable also that the weld contains large amounts of gases, such as nitrogen, and probably hydrogen in the oxy-acetylene process. It would appear that improvements could be made by adding vanadium, manganese, or titanium, or combinations of these elements, to the welding rod. High manganese rod (about 0.60 per cent.), and vanadium rod (about 0.15 per cent.) do make cleaner oxy-acetylene welds, but the difficulty of welding is increased by the slag formed, especially with manganese. It is difficult to float this slag to the surface, and this must be done to make the weld free from inclusions and laps.

The agitation of the melted metal above referred to, and the melting of the original material, which is usually higher in carbon than the welding rod, cause a mixture of the high- and low-carbon materials, especially near the edge of the weld. There is also quite an amount of carbon burned out during the melting; so that there is a change in the carbon content of the metal from the original toward the center of the weld. In addition, there is a great variation in the heating and cooling rates, and in the chemical composition of the welding rods used in different shops, so that great differences in structures of welds may be expected. It should also be remembered that fusion welding is not yet an exact process, and probably never will be, although much better and more uniform results may be obtained with proper care and manipulation than at present.

The above variations result in a variety of structures whose appearances are shown in Fig. 9 to 48. Fig. 9 and 10 show the same apparent structure of the cementite as appears in low-carbon steel wire. This is also seen in Fig. 12, 13, 14, and 20, which are oxy-acetylene welds; and in Fig. 21 and 22, which are electric welds. The weld in Fig. 9 was made with wrought-iron wire, in 10 with low-carbon steel wire, and in 13, 14, and 20 with wrought-iron wire. There is a trace of this structure in Fig. 11, which was made with vanadium-steel wire of about 0.15 per cent. carbon and 0.15 per cent. vanadium. The weld in Fig. 12 was made with wire containing about 0.15 per cent. carbon and 0.50 per cent. manganese. The weld in Fig. 21 and 22 was made with ordinary steel wire, the composition of which is not known.

All of these photographs, except 14, are taken from specimens etched with picric acid. Fig. 14 shows sodium picrate etching, and the structure is somewhat pearlitic in appearance, although, with picric acid etching, the appearance is the same as in Fig. 20. At higher magnifications, these spots appear as in Fig. 15 to 19 inclusive and Fig. 28. There is little difference in the appearance of electric and oxy-acetylene welds; in both cases, the particles are partly darkened only. The author has not been able to get them entirely dark, or to resemble Fig.

8. They all, however, darken more or less, sometimes around the edges, sometimes in the center. With picric acid etching, they are brilliant, and the resemblance to Fig. 6 is marked.

The location of the spots is usually at the grain boundaries, as in ordinary steel; but many times they appear to penetrate the grains, usually parallel to each other, as in Fig. 19, as a general rule extending to the grain boundaries, but never passing from one grain to another. This parallelism, especially of the elongated particles, can be noticed in Fig. 26, 27, and 31. These are all electric welds, and these bright elongated particles appear much more frequently, from the writer's experience, in electric welds than they do in oxy-acetylene welds. They are, however, present in some oxy-acetylene welds, particularly those which are rapidly cooled, as shown in Fig. 15 and 16.

The reason for this peculiar appearance of what the writer has called cementite, is not known to him. It is quite difficult to darken these spots with sodium picrate, and it requires quite a long boiling, some-

| Specimen | Made with | Original | Process | Remarks |
|---|---|---|---|---|
| *Q*3 | Steel wire | Steel plate | Electric | Weld |
| *R* wire | Wire *D* | Yes | | |
| *Q*14 | Wire *D* | | Oxy-acetylene excess oxygen | Button |
| *Q*15 | Wire *D* | Steel plate | Electric | Weld |
| 116 | Steel wire | Steel plate | Electric | Weld |
| *N*1 | Wire *C* | Bar machine steel | Electric | Weld |
| *LG* | Wrought iron wire | Steel plate | Oxy-acetylene | Weld |
| *RI*2 | Wrought iron wire | Bar machine steel | Oxy-acetylene | Weld |
| *RS*2 | Wire *D* | Bar machine steel | Oxy-acetylene | Weld |
| *RV*2 | Wire *A* | Bar machine steel | Oxy-acetylene | Weld |
| 23 | Wire *B* | .............. | Oxy-acetylene | Button |
| M.V. Sp. 9 | Wrought iron wire | 0.77*C* steel | Oxy-acetylene | Weld |
| 167*X* | Steel wire | V-steel plate, 0.23*C* | Electric | Weld |
| *NW*1 | Wire *C* | | Oxy-acetylene | Button |

| | Wire *A* | Wire *B* | Wire *C* | Wire *D* |
|---|---|---|---|---|
| Carbon.............. | 0.160 | 0.13 to 0.18 | Shown in | not over 0.080 |
| Sulphur.............. | 0.034 | Max. 0.045 | Fig. 29 | not over 0.025 |
| Phosphorus.......... | 0.015 | Max. 0.050 | .......... | not over 0.020 |
| Manganese........... | 0.190 | 0.40 to 0.60 | .......... | not over 0.150 |
| Silicon.............. | 0.110 | .......... | .......... | not over trace |
| Vanadium......... | 0.190 | | | |

No further information as to composition of wire is at hand than given above, which are specifications and not actual analyses. "Button" means that wire was melted down into a button weighing about 25 grams, and cooled in air.

times 45 min., before the spots can be seen. They appear only in welds made with comparatively low-carbon material, and the lower the carbon, the less easy it is to darken them. For instance, Fig. 23, 24 and 25 were made from specimen *Q*15, which was first etched very lightly with picric acid to remove part of the film produced by polishing. The result is shown in Fig. 23. Unfortunately, the weld is so full of oxide that it is difficult to see the effect of the etching. The piece was then etched in sodium picrate for 20 min. The result is shown in Fig. 24, and is practically negligible. It was then etched again in sodium picrate for 20 min., with the result that the spots were somewhat darkened, quite clearly so under the microscope. The piece was then etched in picric acid again without polishing, and the result is shown in Fig. 26.

At higher magnification, Fig. 27 shows in one spot a somewhat pearlitic structure, but the rest of the particles do not appear to show it. Whatever their nature, it would appear that the structure is not a satisfactory one in steel. The spots are evidently harder than the metal surrounding them; and the question of proper heat treatment for their elimination appears to be important, because they seem to be more or less carburized, and at times, even in oxy-acetylene welds, lie in a line parallel with the line of the weld, and close to it; and also because it has been the writer's observation that many times they exist where there is a poor union between the original and added materials, as in Fig. 13. Nothing has been done in this connection to the writer's knowledge, but he hopes to be able to carry out some tests.

One of the most interesting structures found in welds made by both methods (but never in a properly made oxy-acetylene weld, as far as the writer has observed), and without exception in electric welds, is that shown in Fig. 22 and 34 to 45 inclusive. The first time it was seriously noticed by the writer was when he observed the field shown in Fig. 42. This specimen (167*X*) is from an electric weld, probably made with ordinary steel wire, although this is not certain. What attracted his attention was the peculiar relative position of the lines in the center of the field. Their relation is better seen in Fig. 43, taken at higher power. It will be noticed that these lines are parallel to the sides and diagonals of a square, and to the lines drawn from the corners of a square to the middles of the opposite sides.

The writer had been reading Professor Howe's book and he had noticed the statement[4] that such lines would represent the intersections of the face planes of a 211 trapezohedron with a cube face. This statement is made in connection with the subject of Neumann lines. An examination of other welds showed that these lines were prominent in all electric welds. They do not always, of course, assume the relation shown in

[4] H. M. Howe: *The Metallography of Steel and Cast Iron.*, 426, ¶ 590. N. Y., McGraw-Hill Book Co., 1916.

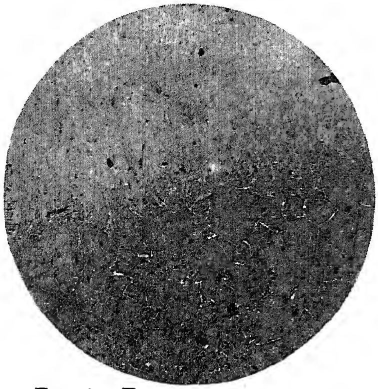

FIG. 1.—ELECTRIC WELD, SHOWING COARSENING OF GRAIN IN ORIGINAL MATERIAL. SAME MATERIAL AND THICKNESS AS FIG. 2. ON LINE OF WELD. ETCHED PICRIC ACID. × 37.

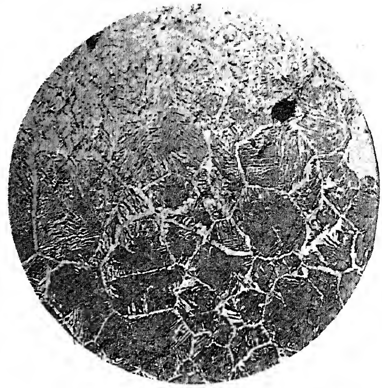

FIG. 2.—OXY.-ACET. WELD, SHOWING COARSENING OF GRAIN OF ORIGINAL. SAME MATERIAL AND THICKNESS AS FIG. 1. ON LINE OF WELD. ETCHED PICRIC ACID. × 37.

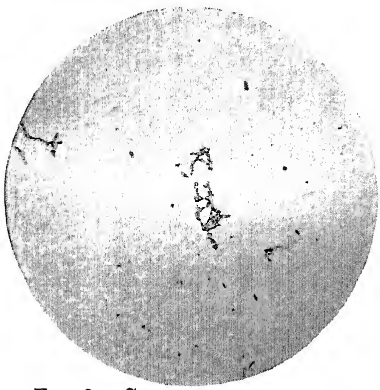

FIG. 3.—CEMENTITE SPOT IN LOW-CARBON STEEL WIRE. ETCHED SODIUM PICRATE 10 MIN. × 148.

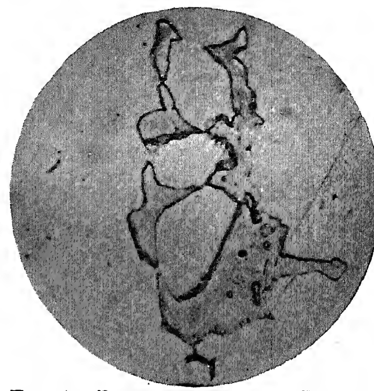

FIG. 4.—PART OF SPOT IN FIG. 3. × 887.

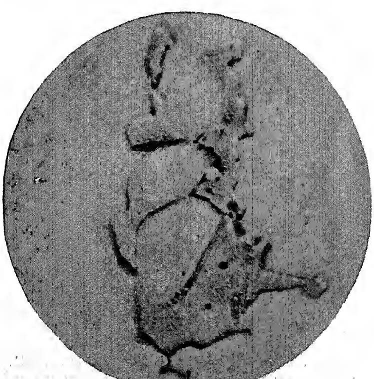

FIG. 5.—SAME AS FIG. 4, BUT WITH LIGHT SLIGHTLY TURNED TO ONE SIDE. × 887.

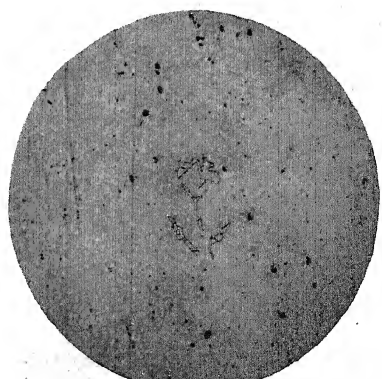

FIG. 6.—CEMENTITE SPOT IN LOW CARBON STEEL WIRE. ETCHED PICRIC ACID. × 148.

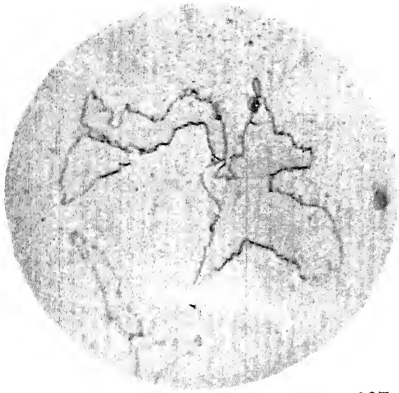

FIG. 7.—SAME AS FIG. 6, BUT × 887.

FIG. 8.—CEMENTITE SPOT IN LOW-CARBON STEEL WIRE. ETCHED 45 MIN. SODIUM PICRATE. × 525.

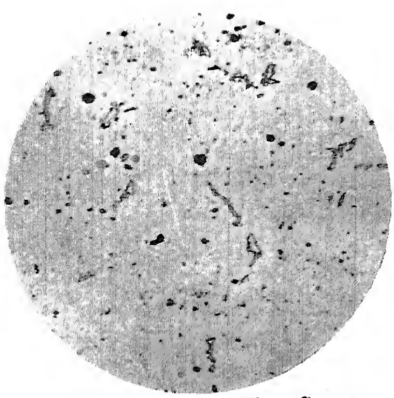

FIG. 9.—SPECIMEN *RI*2. SPOTS OF CEMENTITE AND OXIDE DOTS. IN WELD. ETCHED PICRIC ACID. × 317.

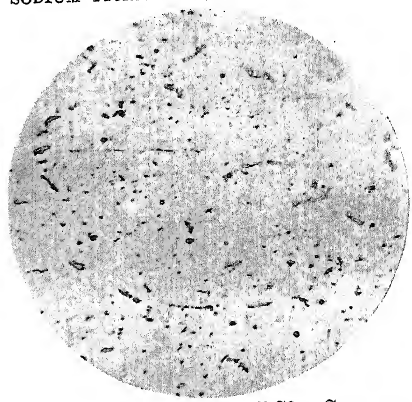

FIG. 10.—SPECIMEN *RS*2. SPOTS OF CEMENTITE AND DOTS OF OXIDE. IN WELD. ETCHED PICRIC ACID. × 317.

FIG. 11.—SPECIMEN *RV*2. PEARLITE, CEMENTITE SPOTS AND SOME OXIDE DOTS. IN WELD. ETCHED PICRIC ACID. × 317.

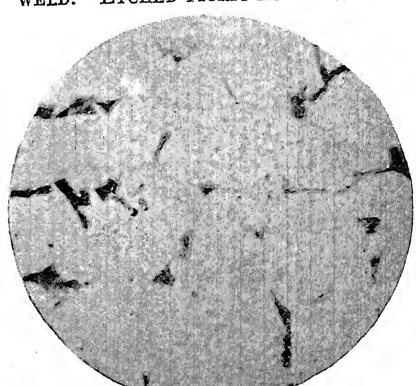

FIG. 12.—SPECIMEN 23. PEARLITE, CEMENTITE SPOTS, AND VERY LITTLE OXIDE. IN WELD. ETCHED PICRIC ACID.

FIG. 13.—SPECIMEN *LG*. CEMENTITE SPOTS AND PEARLITE. IN ORIGINAL JUST NEXT TO WELD. WHITE DOTTED LINE, LINE OF WELD. ETCHED PICRIC ACID. × 317.

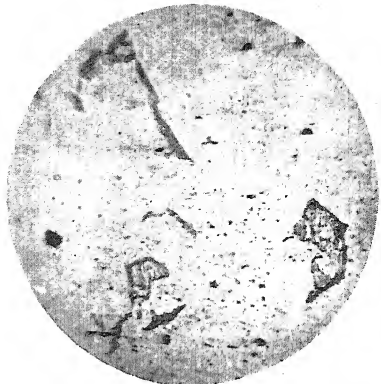

FIG. 14.—SPECIMEN *LG*. CEMENTITE SPOTS. IN WELD JUST NEXT TO ORIGINAL. ETCHED SODIUM PICRATE. × 317.

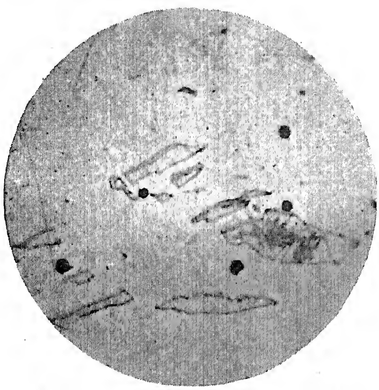

FIG. 15.—SPECIMEN *Q*14. CEMENTITE SPOTS PARTLY DARKENED. ETCHED SODIUM PICRATE. PART OF PHOTO 37. × 887.

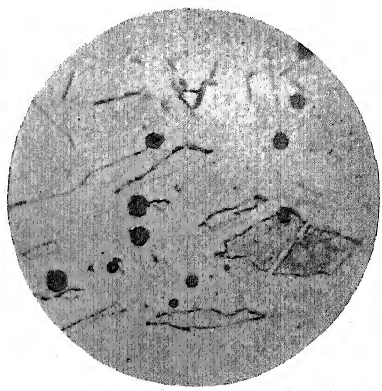

FIG. 16.—SPECIMEN *Q*14. SAME FIELD AS PHOTO 15, BUT REGROUND. ETCHED PICRIC ACID. PART OF PHOTO 39. × 887.

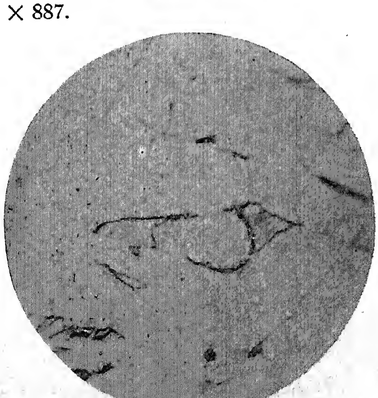

FIG. 17.—SPECIMEN *Q*14. ANOTHER CEMENTITE SPOT. ETCHED SODIUM PICRATE. × 887.

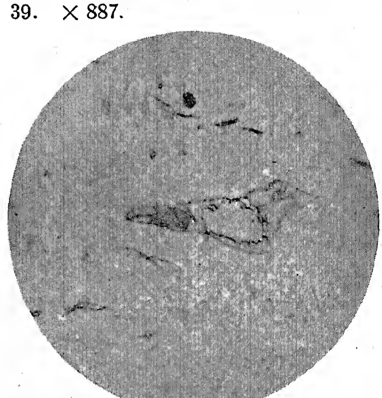

FIG. 18.—CEMENTITE SPOT. SAME AS FIG. 17, BUT REGROUND. ETCHED PICRIC ACID. × 887.

FIG. 19.—SPECIMEN $Q14$. LOCATION OF LARGER SPOTS OF CEMENTITE. ETCHED PICRIC ACID. × 887.

FIG. 20.—SPECIMEN $LG$. ALONG LINE OF WELD. LINE OF CEMENTITE AND PEARLITE SPOTS. ETCHED PICRIC ACID. × 317.

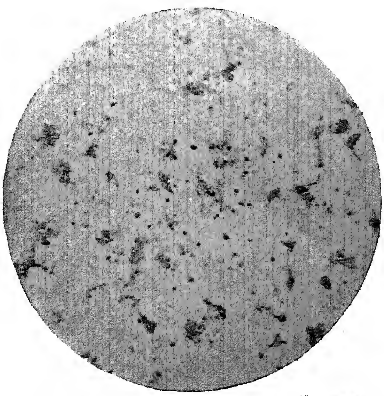

FIG. 21.—SPECIMEN $Q3$. IN WELD JUST NEXT TO ORIGINAL. CEMENTITE AND PEARLITE SPOTS. × 317.

FIG. 22.—SPECIMEN $Q3$. IN BODY OF WELD. CEMENTITE AND PEARLITE SPOTS, OXIDE, AND LINES. × 317.

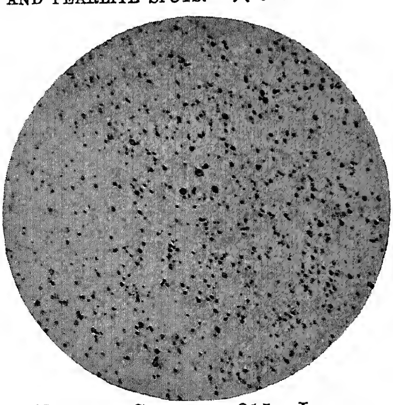

FIG. 23. SPECIMEN $Q15$. IN WELD. CEMENTITE VERY FAINT. ETCHED VERY LIGHTLY, PICRIC ACID. WIRE CONTAINS ABOUT 0.09 PER CENT. $C$. × 317.

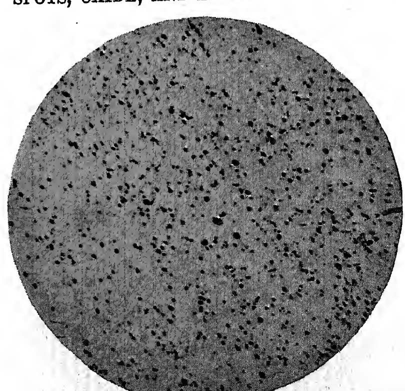

FIG. 24.—SPECIMEN $Q15$. SAME FIELD AS FIG. 23, ETCHED SODIUM PICRATE AFTER PICRIC ACID. VERY LITTLE DARKENING OF CEMENTITE. × 317.

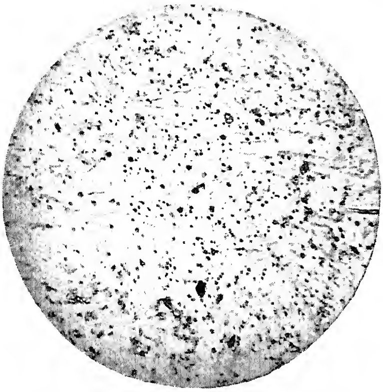

FIG. 25.—SPECIMEN $Q15$. SAME AS FIG. 24, BUT FURTHER ETCHED SODIUM PICRATE. TOTAL ETCHING TIME 40 MIN. SPOTS SOMEWHAT DARKER. $\times$ 317.

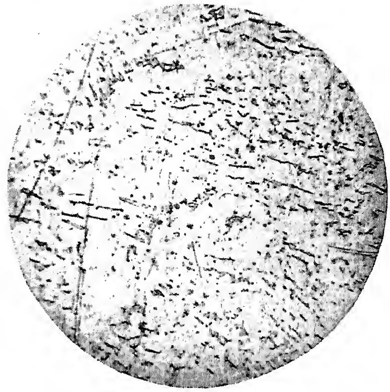

FIG. 26.—SPECIMEN $Q15$. SAME AS FIG. 25, BUT ETCHED WITH PICRIC ACID. GENERAL STRUCTURE OF WELD. $\times$ 317.

FIG. 27.—SPECIMEN $Q15$. SAME AS FIG. 26, BUT $\times$ 887.

FIG. 28.—SPECIMEN $N1$. CEMENTITE SPOTS DARKENED BY ETCHING WITH SODIUM PICRATE. $\times$ 887.

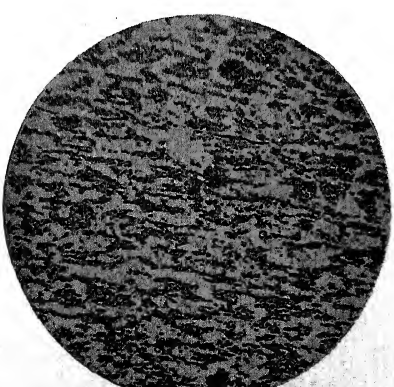

FIG. 29.—WIRE USED TO MAKE WELD $N1$, AS RECEIVED. ETCHED PICRIC ACID. $\times$ 148.

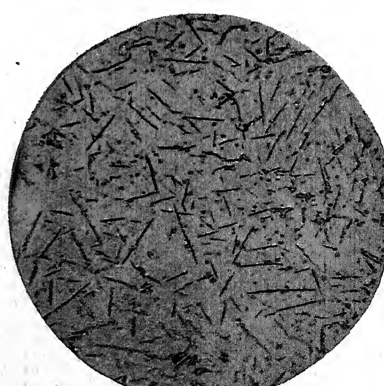

FIG. 30.—SPECIMEN $N1$. CEMENTITE SPOTS AND LINES. ETCHED PICRIC ACID. $\times$ 317.

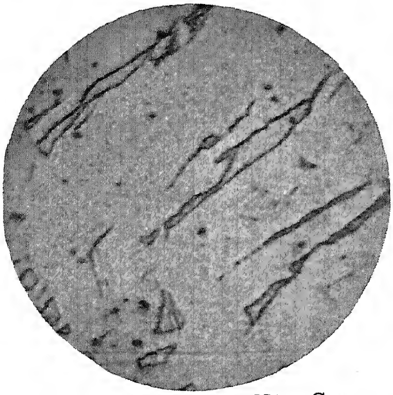

FIG. 31.—SPECIMEN $N1$. CEMENTITE SPOTS. FIELD JUST SOUTHWEST OF FIG. 30. ETCHED PICRIC ACID. × 887.

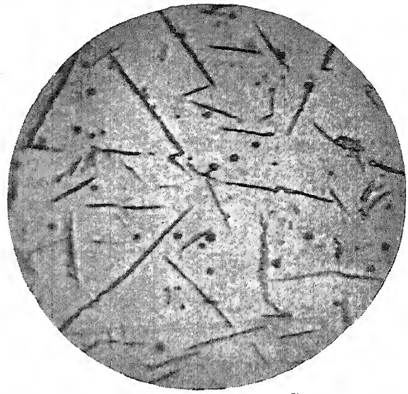

FIG. 32.—SPECIMEN $N1$. SAME FIELD AS FIG. 30, BUT × 887.

FIG. 33.—SPECIMEN $N1$, SHOWING LINES. ETCHED 45 MIN. SODIUM PICRATE. × 317.

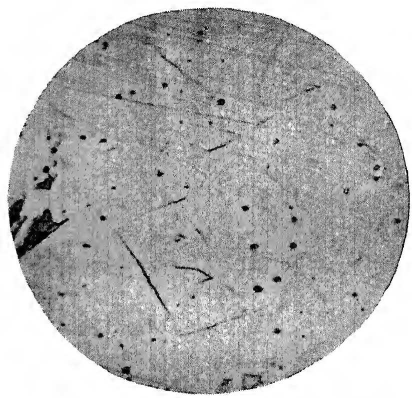

FIG. 34.—SPECIMEN $N1$. SAME AS FIG. 33, BUT × 887.

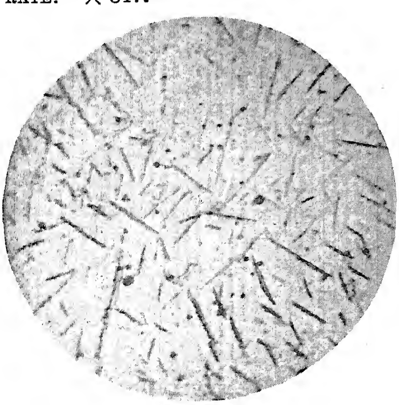

FIG. 35.—SPECIMEN 116. ETCHED PICRIC ACID. SHOWS LINES IN ELECTRIC WELD. × 887.

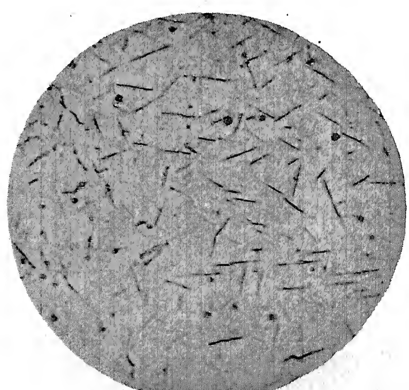

FIG. 36.—SPECIMEN $MVS9$, SHOWING LINES IN $OA$ WELD, JUST NEXT TO ORIGINAL $0.77C$ STEEL. ETCHED PICRIC ACID. × 317.

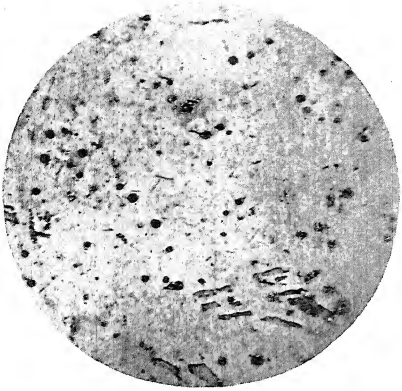

FIG. 37.—SPECIMEN $Q14$. CEMENTITE SPOTS AND LINES DARKENED BY SODIUM PICRATE ETCHING. $\times$ 317.

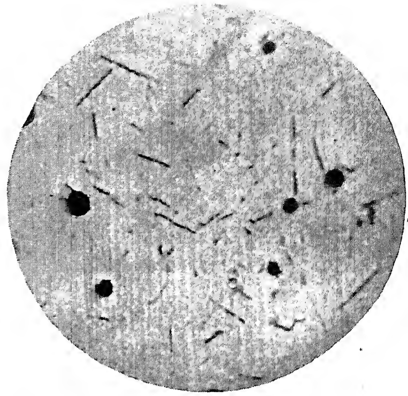

FIG. 38.—SPECIMEN $Q14$. SAME AS FIG. 37, BUT $\times$ 887.

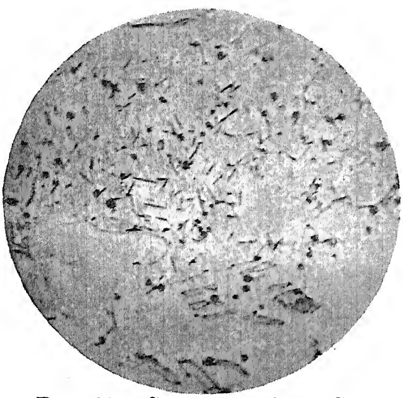

FIG. 39.—SPECIMEN $Q14$. SAME FIELD AS FIG. 37, AFTER REGRINDING AND ETCHING WITH PICRIC ACID.

FIG. 40.—SPECIMEN $Q14$. SAME AS FIG. 39, BUT $\times$ 887.

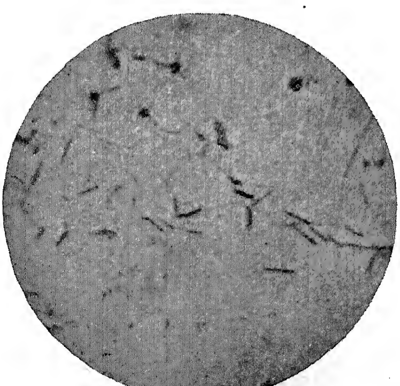

FIG. 41.—SPECIMEN $Q14$. LINES ENDING AT GRAIN BOUNDARIES. ETCHED PICRIC ACID. $\times$ 887.

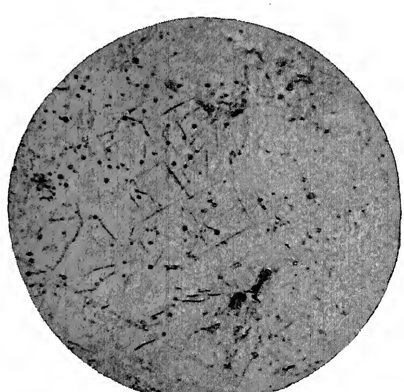

FIG. 42.—SPECIMEN $167X$. SHOWING LINES. ETCHED PICRIC ACID. $\times$ 317.

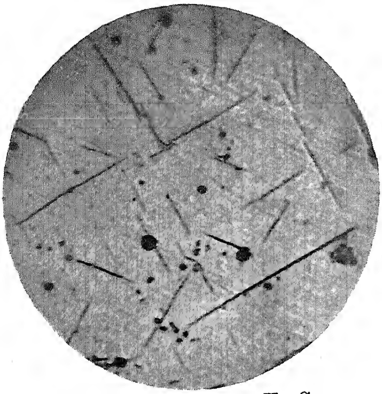

FIG. 43.—SPECIMEN 167$X$. SAME AS FIG. 42, BUT $\times$ 887.

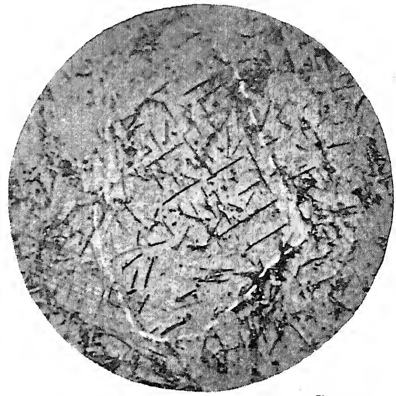

FIG. 44.—SPECIMEN 167$X$. SAME AS FIG. 42, BUT FURTHER ETCHED IN PICRIC ACID.

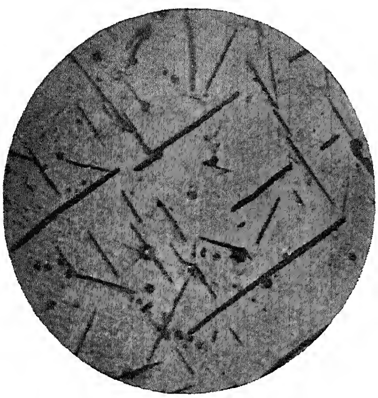

FIG. 45.—SAME AS FIG. 43, BUT MORE HEAVILY ETCHED

FIG. 46.—SPECIMEN $NW1$. LINES IN CENTER GRAIN. ETCHED NITRIC ACID. $\times$ 317.

FIG. 47.—SPECIMEN $RV2$. PEARLITE LINES IN GRAIN. ETCHED NITRIC ACID. $\times$ 317.

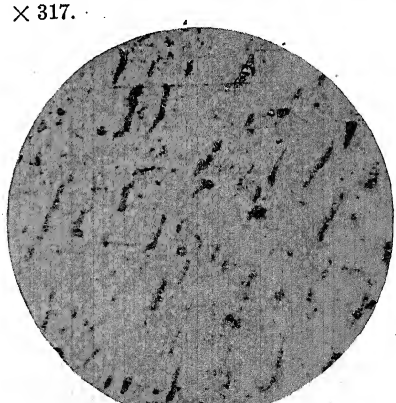

FIG. 48.—SPECIMEN $RV2$. PEARLITE SPOTS. ENTIRE FIELD IS ONE GRAIN. ETCHED NITRIC ACID. $\times$ 317.

Fig. 43, but appear frequently as in Fig. 22, 30, 32, 35, 39, and 40. They do not extend beyond the grain boundaries as shown in Fig. 44, where the approximate outline of the grain is shown by the white dotted line. This is also shown in Fig. 41, and somewhat indistinctly at the upper part of Fig. 16.

Up to that time, they had never been observed by the author in oxy-acetylene welds, but shortly after that, the field shown in Fig. 36 was noticed in a weld made in 0.77 per cent. carbon steel where the added material was wrought-iron wire. In this case, the location of the lines was confined to a very narrow zone just next to the original material; while in electric welds made with wire containing considerable carbon, such as shown in Fig. 29, the weld is full of them, no special location being noticed. In welds made with low-carbon wire, they seem to select the places where the cooling is the most rapid. This is also true in oxy-acetylene welds. In both the latter cases, they are uneven and erratic in their distribution. It does appear, however, that welds made with higher-carbon material contain more of the lines than those made with wire containing less carbon. Fig. 30 and 35 are illustrations of welds made with higher-carbon material than in specimen *Q*15, which was made with wire containing not over 0.10 per cent. carbon. This wire is shown in Fig. 3 and 6, the lines in *Q*15 being infrequent. In specimen *Q*14, which was made by the oxy-acetylene process, using a strongly excess-oxygen flame, the lines were found in only two places. These were at the edge of the piece, where it would cool rapidly. The center was entirely free from them. The material used to make this specimen was low-carbon steel wire.

The author has found three references to these lines.[5] In the first reference, they are believed to be oxide; in the third, they are called magnetic oxide; and in the second, they are stated to be cracks or fissures (Spalten). He believes that none of these explanations is correct, and is strongly inclined to believe that they are very minute plates of cementite rejected along certain crystallographic planes. His reasons for this are as follows:

*First.*—They darken when boiled in sodium picrate. This is shown in Fig. 16, 33, 34, 37 and 38.

*Second.*—They are more numerous with higher-carbon wire than with low.

*Third.*—When heated above the critical range, say to 750° C., for 5 min., and allowed to cool in the furnace from 750° C. to 710° C. at the

[5] E. Höhn: Tests of Autogenous Welds of Boiler Plate. *The Locomotive*, published by The Hartford Steam Boiler Insurance Co. (October, 1916 and January, 1917), **31**, No. 4 and No. 5, 122. See also *Mitteilungen über Forschungsarbeiten auf dem. Gebiete des Ingenieurwesens, Verein deutscher Ingenieure* (1910), **83** and **84**, 18–20.

Law, Merritt and Digby: *Op. cit.*, 114 and Plates 19 and 20.

rate of 7° per minute; from 710° C. to 695° C. at the rate of 3° per minute; and from 695° C. to 635° C. at the rate of 8° per minute, the lines disappear. (This is due to a suggestion of Professor Howe.)

Other tests showed that a considerably quicker cooling rate, an average between the above limits of 14° per minute, was also sufficient. It would appear to the author that neither cracks nor oxide would meet the above conditions.

The writer has wondered if there is not some connection between these lines and those of free cementite in hyper-eutectoid steel. In Howe's book, previously mentioned, a number of photographs show the same structure; *i.e.*, the same relative position of lines. On Plate 3, Fig. *I*, the center grain shows them. In Fig. *G*, the bottom of the grain shows lines almost identical with those in Fig. 42. Plate 9, Fig. *I*, has a small, roughly hexagonal grain at the bottom and one toward the upper right-hand corner where the structure is shown rather dimly. There is a large grain in Fig. *K*, Plate 9, which shows the same thing. Fig. *F*, Plate 9, has a grain toward the bottom of the center in which the same directions appear.

Sauveur gives a very clear photograph in which the large center grain shows the general directions.[6]

It also appears that under certain conditions in welds, particles that are evidently pearlite may assume the same directions. Fig. 46, from a specimen made by melting the wire shown in Fig. 29, shows parallel plates of pearlite structure in the center grain, the grain boundaries being clearly defined. Fig. 47 shows two directions of pearlitic masses in the same grain. Fig. 48 shows pearlitic spots extending in two directions, although mostly in one. The grain, from which this photograph was taken, more than covers a 4 by 5 plate, and the directions were the same in the entire grain.

There would also appear to be some connection between these lines and those found in Andrews' experiments on iron and nitrogen,[7] Plate 1, Fig. 1, showing lines of the same appearance. The grain boundaries also appear as dark lines of the same width, which is not so in welds, in which even heavy etching shows nothing more than is usual in steel.

In a paper on corrosion in steam boiler plates,[8] Fig. 22, 23, and 25,

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[6] A. Sauveur: *The Metallography and Heat Treatment of Iron and Steel*, 129. Cambridge, Mass., 1916.

[7] J. H. Andrew: Influence of Gases upon the Critical Ranges of the Iron-Carbon Alloys. *Carnegie Scholarship Memoirs* (1911), **3**, 236–248. See also paper by M. H. Braune: Influence de l'azote sur le fer et l'acier. *Revue de Métallurgie, Mémoires* (1905), 497–502, and note by Le Chatelier, in which he states that these lines are characteristic of brittle iron and are probably due to hydrogen or nitrogen.

[8] Ch. Frémont and F. Osmond: Sillons de corrosion dans les toles de chaudières à vapeur. *Revue de Métallurgie, Mémoires* (1905), 775.

page 780, similar lines are shown clearly. Of these lines, the authors say "If, then, we are dealing with a brittle metal, iron or steel, repeated bendings create in this metal grooves and Neumann lamellæ. These crystalline deformations, lines of most easy attack, open ways to the corrosion." Again they say, speaking of the photograph on page 782, "Sheet of iron D, brittle. Section parallel and close to the surface, cracks and Neumann lines."

There appears to be no question that in the cases cited, the lines are due to nitrogen and corrosion. But their similarity in appearance does not prove that the lines found in welds are due to either of these causes.

Howe also shows,[9] after Ewing and Humfrey, Fig. *C*, Plate 25, in which the grain between grains 3 and 11 shows three directions of slip bands, which are parallel to the sides and diagonal of a square. He also shows on Plate 21, Fig. *H*, four directions of slip bands in a specimen of Invar, which are parallel to the sides and diagonal of a square and to a line drawn from its corners to the center of the opposite side.

One peculiarity of these lines in welds is that they widen considerably by prolonged etching. Fig. 44 and 45 show this when compared with Fig. 42 and 43. Under the microscope, they appear to resemble etching pits in that they are deeper in the center than at the sides, and this depth appears to increase as the etching is prolonged. They also have the appearance of being stepped when the light is thrown by the reflector in the right direction.

Howe speaks of this widening of Neumann lines by etching,[10] although his Fig. 100 does not show steps. To the writer these steps appear quite clear, although it is exceedingly difficult to photograph them because of the high power required and the fact that the steps are not in the same planes. It is possible that still further etching would enable the steps to be seen at lower power, but this has not been tested.

There is also, in many cases, a very faint white line which can be seen in Fig. 45 in the center of two of the lines and in Fig. 35. It is possible that this is the edge of a plate of cementite, although it may be due to an optical effect. The writer is inclined to the former opinion.

It would appear to the writer that, owing to the similarity in appearance under different conditions, in low-carbon and high-carbon steels —from their resemblance, at least occasionally, to slip bands—to Neumann lines at least in direction and to lines undoubtedly due to nitrogen and to corrosion there must be an intimate relationship between these various kinds of lines. He would suggest that, as a cleavage plane is a plane of weakness, Neumann lines are lines of incipient fracture along these weak planes; that the rejection of free cementite to the planes it occupies,

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[9] H. M. Howe: *Metallography of Steel and Cast Iron*.

[10] *Op. cit.*, 419.

and the location of the lines caused by nitrogen and corrosion, are because of the inherent weakness of these planes, which weakness would seem to him manifested by the appearance of slip bands in the same directions.

The author has ventured to call attention to these peculiar structures for two reasons:

*First.*—Because of their probable practical importance in connection with the strength of welds.

*Second.*— Because of their probable theoretical importance in connection with the deformation of iron and steel.

The latter being a subject of which he knows nothing, he allows the matter to rest, trusting that his observations may be of some interest and possible value to those competent to deal with the subject.

## Discussion

H. M. Boylston, Cambridge, Mass.—We should feel greatly indebted to Mr. Miller for collecting this admirable series of photographs, but I cannot quite agree with his conclusions as to their significance. He apparently concludes that these needle-like constituents which he has exhibited in a good many cases in material welded by both processes, are cementite; that is, as we understand it, a carbide of iron. Now there seem to be several discrepancies in his proof that those are cementite, and I would like to point out a few of them.

In the first place, most of the photographs in his paper show these as dark needles on a light background, apparently on a background of free ferrite, and whether those specimens are etched with sodium picrate or picric acid, the lines are still dark. Now we know that free cementite grains remain perfectly bright when picric acid is used as an etching medium, yet some of his figures, etched with picric acid, show a darkening of supposed cementite.

The second point is that sodium picrate etching is not sufficient in itself as a test. When cementite contains a good deal of manganese, the cementite is not necessarily darkened. My experience has been that a series of iridescent colors are produced, instead of the usual darkening, in spiegeleisen, for instance, whereas some of the cementite is left fairly bright. Mr. Comstock showed a year or two ago that sodium picrate would remove sulphide particles in steel, giving a black appearance instead of the usual dove-gray color, and that is one of our principal methods of determining whether the sulphide is present. Therefore the mere fact that a constituent darkens with sodium picrate is not conclusive proof that it is cementite. Of course, I do not mean to say that any of these needle-like forms look like sulphide. I found lines which looked like those shown in Fig. 45, in apparently carbon-free iron produced by the thermit process.

Another curious thing is that in Mr. Miller's sample this supposed cementite exists in the needle form, apparently on the background of free ferrite. We used to think that free ferrite and free cementite could not exist together in steel. That idea has been proved false long ago; but when they do occur together, the cementite is found in globule form. That does not prove that it cannot exist in the needle form, but it seems to me very unlikely, from the laws we know, that free cementite could exist in needle form in the presence of nothing else than free ferrite. Anything occurring in the needle-like form must apparently exist in the cleavage planes of the crystals.

I could show Mr. Miller, in a great many steel castings, free ferrite in the same position. Low-carbon steel castings quite often show the parallelogram structure of free ferrite; also hyper-eutectoid steel castings generally show a parallelogram structure of free cementite. Hence, the direction of these lines, as brought out, is merely an indication that the crystals are of geometric form and have cleavage planes in those several directions. I do not think that the position of those lines is any proof that they are cementite. In fact, on page 702, Mr. Miller says that "it seems probable also that the weld contains large amounts of gases, such as nitrogen, and probably hydrogen from the oxy-acetylene process." Without myself studying it very closely in an experimental way, I should say that he proved pretty conclusively that these needles are connected with the presence of nitrogen.

On page 713 Mr. Miller says that he made these needles disappear by heating to 750° C., with various rates of cooling. Did Mr. Miller observe any of the constituents remaining in some other form, as, for instance, in the form of granules or little nodules, or rounded areas?

S. W. Miller.—The needles disappeared entirely, although it was impossible to obtain the same field after the piece had been treated. However, the general appearance showed a considerable coalescence of the particles of cementite. In other words, there appeared to be an entire absence of needles, the particles of cementite were more rounded and larger, and in a few cases where there was sufficient cementite, small particles of pearlite appeared.

H. M. Boylston.—It seems to me that we might possibly help to prove the nature of this material by heating some pieces known to contain it, to a sufficient temperature, in a vacuum, to see if we can extract the gases, and then note whether the extraction of the gases would make any difference in the structure, the gases also being analyzed.

W. E. Ruder,* Schenectady, N. Y.—The first idea that struck me on looking over these photographs was that Mr. Miller undoubtedly

* Metallurgist, General Electric Co.

had nitride of iron, because if you have had any experience with that material, and have examined it under the microscope, you cannot fail to recognize it if you see it again. The only objection I can see to the theory that this is nitride is the extreme difficulty with which nitride of iron is formed from nitrogen and iron; it is formed very easily with ammonia and iron, but with nitrogen it is very difficult.

Regarding Mr. Boylston's suggestion of heating these specimens in a vacuum, I have found that, although the amount of nitride is apparently not reduced by ordinary annealing, a short vacuum anneal at 1000° C. does change the appearance of the structure, indicating some reduction in the amount present. I have recently subjected a sample of nitrogenized iron for 3 hr. at 1000° C., in vacuum, and find a decided reduction in the amount of nitride present.

It seems to me that the most plausible explanation of these lines is that they are martensite, formed by the extremely rapid rate of cooling. That this quenching action is very great is indicated by the abnormal hardness of spot welds, even when made of low-carbon stock.

Zay Jeffries,* Cleveland, Ohio.—I have had occasion to study the structures of acetylene welds, as well as arc welds used in various welding materials. Regarding the formation of these needles, I am of the opinion that they are not cementite, for the following reasons:

1. That their etching characteristics are not at all like the etching characteristics of cementite, either with picric acid or other reagents. If they were cementite, they should remain brighter than the surrounding material when etched with picric acid, and darker than the surrounding material when etched with sodium picrate.

2. The formation of the free cementite under the stated conditions disobeys the phase rule so completely that I think we might almost discard the idea of free cementite at once.

I realize, of course, that the formation of cementite globules disobeys certain features of the phase rule, and that the phase rule is not what we want to go by—we want to go by facts—but let us see what those needles would have to do in order to form free cementite under the conditions existing during the cooling of that weld.

Ferrite, not cementite, should separate at the grain boundaries first, and then if diffusion cannot take place rapidly enough, the ferrite should begin to separate in the cleavage planes of the austenite. The cementite would remain in solution in the steel until it concentrated to 0.9 per cent. in carbon, and then should form either pearlite, sorbite, troostite, or martensite, according to the rate of cooling. For the cementite to form in the free needle state under those conditions would be contrary to all

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* Director of Research, Aluminum Castings Co.

laws of phase equilibria now recognized. But we do not need to consider that the needles are free cementite in order to explain their occurrence.

The formation of those needles will vary with the rate of cooling, through the critical range. The acetylene weld cools very much more slowly than the electric weld. That is due to the fact, as Mr. Miller has mentioned, that the acetylene torch heats large adjacent areas of the steel to be welded, whereas in the electric weld a very small portion of the steel is heated; the rest of it remains comparatively cold. The needles occur only occasionally in the acetylene welds I have examined. In the electric welds the needles are most prominent.

Osmond found that steel containing as little as 0.10 to 0.12 per cent. of carbon, when quenched a little above the upper critical range, produced only a little martensite, but when quenched from 1200° to 1300° produced more martensite. We have the condition, in these electric welds, of quenching from the melting point, or 1500°. We have the second condition of a lower carbon content.

If those needles were cementite, the area represented would be sufficient to produce steel of 0.6 per cent. carbon so that their quantity is out of proportion to the quantity of cementite which could form.

We know nothing about the limiting side of martensite from the low-carbon end. I have made welds containing fairly high carbon and would always get martensite, and they are very hard. The electric welds, therefore, cooled fast enough to produce martensite, when there is enough carbon to recognize martensite in its familiar form. If we decrease the carbon content gradually, where is the formation of that martensite going to stop? I cannot say that there is a point between 0.10 and 0.09 per cent. carbon above which we will get martensite, and below which we will not.

I believe that the needle structures (the ones which I have examined at any rate) have a direct bearing on the formation of martensite. The needles are in the same direction and position as the martensitic needles, and all of the needles which Mr. Miller has discussed are in the cleavage planes of the austenite grains. So are martensite needles in the cleavage planes of the austenite, etc. As we increase the grain size of the austenite from which the martensite was formed, we make the needle structure more and more pronounced, but as we decrease the carbon content may we not increase the distance between needles; and if we reduce the carbon very low, may the distance between needles not be something on the order of that which is found in the electric welds?

I have another reason for supposing that those needles relate particularly to the formation of martensite, namely, the material in that condition is harder than it is after being heated above 900° and cooled in the furnace; or, in other words, even though the grain is smaller after grain refinement, the hardness seems to be less.

I wouldn't attempt to say why the martensite looks as it does. Nobody knows what martensite is; nobody knows how many needles, side by side, we must have before we can call it martensite; nobody knows how far apart those needles must be in order to call the material martensite. In other words, most of our work on martensite has been with steels containing more carbon than these steels with which Mr. Miller has been working. There is no reason, however, why the formation of martensite should not be gradual, from zero carbon up to the highest quantity of carbon. The conditions noted by Mr. Miller, namely, the speed of cooling, high temperature previous to cooling, the increased hardness, the position of the needles being the same as the position of the needles of ordinary martensite, and the softening of the weld by annealing, all tend to give one the idea that those needles are of martensite origin.

One thing more about the nitride. Why should any nitride form in the few seconds in which the electric-welded steel is heated to the very high temperature, when oxygen is so abundant? That is, we should have to use up the oxygen before the nitrogen would begin to act; and the evidence is that the oxygen has been quite active. Then, as yet, we have no evidence that the nitride of iron can be broken up by heating it above the $Ac$ point; however, this needle structure can be, and so can martensite.

I have suggested to Mr. Miller that if the needles are martensite they should be broken up by heating below $Ac_3$. This would also be the case if they were lines of deformation. I am of the opinion that if these needles be considered as lines of deformation that martensite needles should be considered as substantially the same thing.

After studying Mr. Miller's micrographs carefully I note that the needles in his electric welds tend to form 60° angles more than 90° and 45° angles. Even where the latter are present, the 60° angles may appear also (see Fig. 42, 41, 36, 35, 34, 25, 22).

George F. Comstock,* Niagara Falls, N. Y. (written discussion†).—I have recently had the pleasure of reading Mr. Miller's interesting paper, and would like to call attention to a reference to this subject which apparently has escaped his notice. It is an abstract of a German research on the subject of nitrogen in steel, and appeared in the *Iron Age* of Feb. 17, 1916, page 432. Photomicrographs of needles such as Mr. Miller found in his welds are given, and their identity is discussed. The conclusion reached is that the needles are slip lines due to the brittleness of the ferrite crystals caused by their absorption of nitrogen. The deformation necessary to develop the slips is supposed to be accomplished either in cutting or polishing the microsections or by cooling strains

* Titanium Alloy Manufacturing Co.

† Received Mar. 30, 1918.

after welding. It seems very reasonable to expect that nitrogen would be absorbed by steel in either electric or oxy-acetylene welding, and the reference I have mentioned gives instances of many unusual structures resulting from the presence of considerable nitrogen with carbon in steel. But the nitrogen is rapidly given off from steel by ordinary annealing in air, hydrogen or vacuum, and this would explain the disappearance of Mr. Miller's needles after annealing. I think the conclusion that the needles are cementite might well be revised in view of this German work on nitrogen.

# The Effect of the Presence of a Small Amount of Copper in Medium-carbon Steel

BY CARLE R. HAYWARD,* AND ARCH. B. JOHNSTON,† S. B., CAMBRIDGE, MASS.

(New York Meeting, February, 1918)

THE effect of copper on steel has been studied by numerous investigators. Before modern testing methods had been developed, blacksmiths noted red shortness in iron, the cause for which was ascribed to the presence of copper. Numerous papers have been published on the corrosion of steels containing various amounts of copper, and a few writers have discussed the effect of copper on the mechanical properties of steel.

The purpose of the investigation described in the following paper was to obtain additional data on the mechanical properties of medium-carbon steel containing small quantities of copper.

Among those who have published results of mechanical tests on copper steels are Ball,[1] who states that copper increases the tensile strength and hardness but lowers the elongation. Stead[2] states that copper steels closely resemble nickel steels containing equivalent percentages of nickel, as regards tensile strength, resistance to shock, corrosion and hardness. Wigham[3] found that copper in small amounts had no injurious effect on steel. Breuil[4] found that with 1 per cent. carbon, copper lessened the brittleness of steel and in low-carbon steel it increased the tensile strength and lowered the ductility slightly. H. H. Campbell[5] states that copper up to 0.25 per cent. slightly raises the elastic limit, elongation and reduction of area.

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[1] E. J. Ball and A. Wingham: On the Influence of Copper on the Tensile Strength of Steel. *Journal of the Iron and Steel Institute* (No. I, 1889), 123–131.

[2] J. E. Stead and J. Evans: Influence of Copper on Steel Rails and Plates. *Journal of the Iron and Steel Institute* (No. I, 1901), **59**, 89–100.

[3] F. H. Wigham: Effect of Copper in Steel. *Journal of the Iron and Steel Institute* (No. I, 1906), **69**, 222–232.

[4] P. Breuil: Copper Steels. *Journal of the Iron and Steel Institute* (No. II, 1907), **74**, 1–78.

[5] *Manufacture and Properties of Structural Steel*, N. Y., Scientific Pub. Co., 1896.

It will be noted that all the above writers agree that copper increases the tensile strength, but they disagree in regard to the ductility. There is little to be found on resistance to shock.

## Materials Used

The steels used were obtained through the kindness of Mr. Frank D. Carney of the Pennsylvania Steel Co. They were furnished in the form of forged bars about 1 in. square in cross-section and of varying lengths. There were three bars of one composition marked No. 41, No. 42 and No. 43 and four bars of another composition marked No. 51, No. 52, No. 53 and No. 54. The chemical analyses furnished with the steel are given in Table 1.

Table 1.—*Analyses of Steels Tested*

| | Nos. 41, 42, 43 | Nos. 51, 52, 53, 54 |
|---|---|---|
| C | 0.380 | 0.365 |
| P | 0.012 | 0.053 |
| Mn | 0.570 | 0.590 |
| S | 0.030 | 0.048 |
| Cu | 0.860 | 0.030 |

It will be noted that the analyses differ only slightly except in P and Cu, and it is probable that the effect of P will be neutralized by the slight difference in C.

Although 0.86 Cu is not generally considered high, for the purpose of designating the steels in this paper, the first will be called high-copper and the second low-copper.

## Preparation of Specimens

The bars were first cut to 7½-in. lengths. The cutting was done with an ordinary reciprocating mechanical saw. Three of the bars gave five specimens each, while the fourth bar, which was somewhat longer, yielded six specimens. This made a total of twenty-one 7½-in. specimens of low-copper steel. Each of the other bars yielded seven specimens, making a total of twenty-one 7½-in. high-copper specimens. The bars were now square, but as this form was not the most favorable for heat treatment, it was decided to turn them down round, on a lathe, in order to obtain the most uniform heating in the furnace. As the threads for the tensile specimens would eventually have to be cut on a ¾-in. round bar, it was decided to turn the 42 specimens down to this size prior to treatment. The steel as forged was fairly soft and turned very easily on the lathe. There was no noticeable difference in the speed of cutting, between the high- and low-copper steels.

## Heat Treatment

The furnace used has been described in a previous paper[6] and the methods employed were the same as used in that investigation.

Three specimens of high-copper and three specimens of low-copper steel were taken for each test and in order to eliminate possible differences in the bars no two specimens in a set were taken from the same bar except in one or two instances. There were seven different heat treatments, and in order to identify the specimens the numbers 1 to 7 were prefixed before the original number of the bars. Table 2 gives the numbers as finally stamped on the specimens.

Table 2.—*Identification of Test Samples*

| Section | High Cu | Low Cu |
|---|---|---|
| (1) | 141, 142, 143 | 151, 152, 153 |
| (2) | 241, 242, 243 | 252, 253, 254 |
| (3) | 341, 342, 343 | 351, 352, 353 |
| (4) | 441, 442, 443 | 452, 453, 454 |
| (5) | 541, 542, 543 | 552, 553, 554 |
| (6) | 641, 642, 643 | 651, 653, 654 |
| (7) | 741, 742, 743 | 751, 752, 754 |

The six bars of section (1) were reserved for tests without heat treatment. The remainder were treated as follows:

The bars under section (2) were heated to 765° C., the switch was pulled, and the final temperature rise registered due to residual heat was 865° C. The bars were allowed to cool in the furnace over night, and removed next morning.

The bars under section (3) were heated to 765° C., the switch was pulled, and the final temperature attained due to residual heat was 860° C. They were then removed and set inclined against a brick on the cement floor to cool in air.

The bars under section (4) were heated to 765° C., the switch was pulled, and the final temperature attained due to residual heat was 845° C. They were removed from the furnace and dropped into buckets of cold water.

The bars under section (5) were heated to 765° C., the switch was pulled, and the final temperature attained was 865° C. They were removed and quenched in water. Then they were replaced and drawn at 360° C., again removed and quenched in water.

[6] Carle R. Hayward: The Effect of Sulphur on Low-Carbon Steel. *Trans.* (1917), **56**, 535.

The bars under section (6) were heated to 765° C., the switch was pulled, and the final temperature attained was 860° C. They were removed and quenched in water; then replaced and drawn at 455° C., and finally removed and quenched in water.

The bars under section (7) were heated to 800° C., by mistake, the switch pulled, and the final temperature attained was 900° C. They were removed, quenched, and then drawn at 580° C., and quenched in water.

## Cutting up the Bars for Tests after Heat Treatment

After heat treatment there were 42 bars ¾ in. in diameter, and 7½ in. long. Thirty-six of these were sawed up into three pieces, 4 in., 2½ in., and 1 in. long. The six bars under section (4) (quenched at 845°) were found to be too hard to saw, and in order to cut them up a thin alundum wheel had to be used. As the wheel was in poor condition, the process took about 4 hr. per bar, so only one high- and one low-copper specimen from this section were cut for tests.

The 1-in. lengths were set aside to be polished on the ends for microscopic work, sclerescope tests, and Brinell hardness numbers.

The 4-in. lengths were turned into standard test specimens 0.505 in. diameter and 2-in. gage length with threaded ends. The bars to be tested as quenched were too hard to turn on a lathe. A 7-in. length was therefore used and the ends annealed in a blacksmith's forge to allow the cutting of the threads. During the heating, the center was kept cool with water. A 2-in. gage length was then ground at the center of the bar.

The 2½-in. lengths were ground down into small rectangular bars 0.395 in. square, and 2 in. long, with a slot cut half way through in the middle of the bar. The slot cutter gave a slot 1 mm. wide.

The machining on both the tensile and shock-test specimens was done through the kindness of the Rhode Island Tool Co.

## Tensile Tests

Each specimen was broken in an Olsen testing machine and readings were taken of its yield point, ultimate strength, reduced diameter and elongation in 2 in. The yield point was determined by watching for the drop of the beam and in a few cases by measuring with calipers the change in elongation under equal increases of load.

The values computed from the observations are given in Table 3. The average results are given in Table 4.

TABLE 3.—*Individual Results of Tensile Tests*

| No. | Yield Point, Lb. per Sq. In. | Ultimate Strength, Lb. per Sq. In. | Reduction of Area, Per Cent | Elongation, Per Cent |
|---|---|---|---|---|
| 143 | 62,900 | 92,300 | 51.8 | 24 5 |
| 143 | 62,400 | 92,900 | 54.5 | 25.5 |
| 141 | 57,500 | 92,600 | 51 8 | 24 5 |
| 151 | 47,300 | 83,600 | 51.8 | 26.0 |
| 151 | 46,200 | 83,400 | 51.8 | 25.5 |
| 154 | 46,200 | 82,700 | 49.0 | 26.0 |
| 242 | 58,300 | 86,500 | 49.0 | 25.5 |
| 243 | 60,500 | 86,500 | 49.0 | 27.0 |
| 241 | 58,800 | 86,600 | 49.0 | 25.5 |
| 252 | 47,100 | 78,100 | 46.2 | 26.0 |
| 253 | 45,200 | 78,200 | 46.2 | 26.5 |
| 254 | 45,200 | 78,100 | 46.2 | 27.5 |
| 351 | 52,500 | 84,000 | 54.5 | 26.5 |
| 353 | 53,400 | 85,300 | 51.8 | 27 0 |
| 352 | 54,000 | 84,800 | 51.8 | 28 5 |
| 341 | 66,400 | 90,200 | 51.8 | 27.5 |
| 342 | 64,400 | 89,900 | 49.0 | 26.5 |
| 343 | 62,100 | 90,900 | 57.3 | 28 0 |
| 551 | 100,000 | 135,000 | 37.1 | 13.5 |
| 553 | 100,000 | 137,900 | 40.2 | 11.0 |
| 554 | 100,000 | 135,800 | 43.3 | 14.0 |
| 541 | 130,000 | 192,500 | 46.2 | 7.5 |
| 542 | 130,000 | 185,300 | 37.1 | 10.5 |
| 543 | 130,000 | 194,000 | 34.0 | 9.5 |
| 651 | 90,000 | 123,900 | 51.9 | 16 0 |
| 653 | 100,000 | 129,800 | 46.2 | 13 5 |
| 654 | 110,000 | 134,200 | 49.0 | 24.5 |
| 641 | 135,000 | 151,400 | 46.2 | 27.5 |
| 642 | 138,300 | 152,000 | 51.9 | 28.0 |
| 643 | 137,500 | 151,000 | 51.9 | 28.0 |
| 752 | 75,000 | 105,800 | 54.5 | 33.0 |
| 754 | 80,000 | 110,000 | 54.6 | 33.0 |
| 751 | 85,000 | 112,900 | 54.6 | 32.0 |
| 741 | 110,800 | 124,000 | 54.6 | 32.0 |
| 742 | 113,700 | 127,200 | 54.6 | 31.5 |
| 743 | 107,400 | 119,800 | 59.8 | 33.0 |
| 453 | 110,000 | 135,900 | 20.5 | 6.5 |
| 443 | ? | 206,800 | | |

TABLE 4.—*Average Results of Tensile Tests*

| Treatment | Yield Point, Lb. per Sq. In. | | Ultimate Strength, Lb. per Sq. In. | |
|---|---|---|---|---|
| | High Cu | Low Cu | High Cu | Low Cu |
| Cooled in furnace | 52,900 | 45,800 | 86,530 | 78,130 |
| Cooled in air | 64,300 | 53,300 | 90,300 | 84,700 |
| Bars as forged | 60,900 | 46,600 | 92,600 | 83,200 |
| Drawn at 580° C. | 110,600 | 80,000 | 123,700 | 109,600 |
| Drawn at 455° C. | 136,900 | 100,000 | 151,500 | 129,300 |
| Drawn at 360° C. | 130,000 | 100,000 | 190,800 | 136,300 |
| Quenched at 845° C. | ? | 110,000 | (207,000)* | 135,900 |
| | Per Cent. Reduction | | Per Cent. Elongation | |
| Cooled in furnace | 49.0 | 46.2 | 26.0 | 26.7 |
| Cooled in air | 52.7 | 52.7 | 27.3 | 27.3 |
| Bars as forged | 52.7 | 50.9 | 24.8 | 25.8 |
| Drawn at 580° C. | 56.3 | 54.6 | 32.2 | 32.7 |
| Drawn at 455° C. | 50.0 | 49.0 | 27.8 | 18.0 |
| Drawn at 360° C. | 39.1 | 40.2 | 9.2 | 12.8 |
| Quenched at 845° C. | .... | 20.5 | .... | 6.5 |

* Broke in threads because of annealing ends for threading.

## SHOCK TESTS

The shock tests were made on the Charpy machine at the Watertown Arsenal. The results are tabulated in Table 5.

## HARDNESS

The hardness of the specimens as determined by the Brinell and scleroscope methods is given in Table 6.

The average results from the Brinell and Charpy tests are plotted in Fig. 1.

## MICROSCOPIC EXAMINATION

The one fact revealed by the microscopic study was that for the same treatment the high-copper steel was finer grained than the low-copper. The quenched and drawn specimens of high-copper steel were also slightly more martensitic.

## DISCUSSION OF RESULTS AND CONCLUSIONS

The results need little interpretation. The table of tensile strengths shows a striking superiority of the high-copper steel. The yield point

TABLE 5.—*Results of Shock Tests*

| No. of Specimen, Copper | | Breaking Force, Ft-Lb per Sq. In. | Average |
|---|---|---|---|
| 141 | High | 282 | |
| 143 | | 279 | 288 |
| 143 | | 302 | |
| 151 | Low | 250 | |
| 151 | | 227 | 246 |
| 154 | | 262 | |
| 241 | High | 233 | |
| 242 | | 244 | 240 |
| 243 | | 242 | |
| 252 | Low | 177 | |
| 253 | | 168 | 173 |
| 254 | | 93(*a*) | |
| 341 | High | 325 | |
| 342 | | 318 | 327 |
| 343 | | 338 | |
| 351 | Low | 236 | |
| 352 | | 258 | 242 |
| 353 | | 232 | |
| 541 | High | 148 | |
| 542 | | 127 | 128 |
| 543 | | 108 | |
| 552 | Low | 175 | |
| 553 | | 91 | 127 |
| 554 | | 115 | |
| 641 | High | 287 | |
| 642 | | 252 | 254 |
| 643 | | 222 | |
| 651 | Low | 233 | |
| 653 | | 220 | 231 |
| 654 | | 240 | |
| 741 | High | 357 | |
| 742 | | 323 | 358 |
| 743 | | 394 | |
| 751 | Low | 268 | |
| 752 | | 284 | 285 |
| 754 | | 304 | |

(*a*) Not averaged in because of visible defect in specimen.

TABLE 6.—*Results of Hardness Tests*

| No. of Specimen, Copper | | Brinell No. | Average | Scleroscope No. | Average |
|---|---|---|---|---|---|
| 141 | High | 179 | | 18 | |
| 143 | | 177 | | 23 | |
| 143 | | 173 | 176 | 19 | 20 |
| 151 | Low | 149 | | 16 | |
| 151 | | 149 | | 17 | |
| 154 | | 149 | 149 | 16 | 16 |
| 243 | High | 158 | | 18 | |
| 242 | | 158 | | 17 | |
| 241 | | 159 | 158 | 18 | 18 |
| 252 | Low | 137 | | 16 | |
| 253 | | 134 | | 15 | |
| 254 | | 134 | 135 | 15 | 15 |
| 341 | High | 170 | | 18 | |
| 343 | | 171 | | 18 | |
| 342 | | 171 | 171 | 19 | 18 |
| 351 | Low | 150 | | 15 | |
| 352 | | 152 | | 16 | |
| 353 | | 156 | 153 | 16 | 16 |
| 441 | High | 444 | 444 | 45 | 45 |
| 454 | Low | 241 | 241 | 26 | 26 |
| 542 | High | 346 | | 40 | |
| 543 | | 364 | | 40 | |
| 541 | | 339 | 350 | 38 | 39 |
| 554 | Low | 241 | | 31 | |
| 553 | | 248 | | 31 | |
| 552 | | 242 | 243 | 26 | 29 |
| 643 | High | 309 | | 35 | |
| 642 | | 313 | | 35 | |
| 641 | | 311 | 311 | 35 | 35 |
| 651 | Low | 232 | | 26 | |
| 653 | | 236 | | 27 | |
| 654 | | 232 | 233 | 26 | 26 |
| 742 | High | 259 | | 30 | |
| 741 | | 254 | | 29 | |
| 743 | | 248 | 254 | 29 | 29 |
| 751 | Low | 215 | | 24 | |
| 754 | | 212 | | 24 | |
| 752 | | 206 | 211 | 23 | 24 |

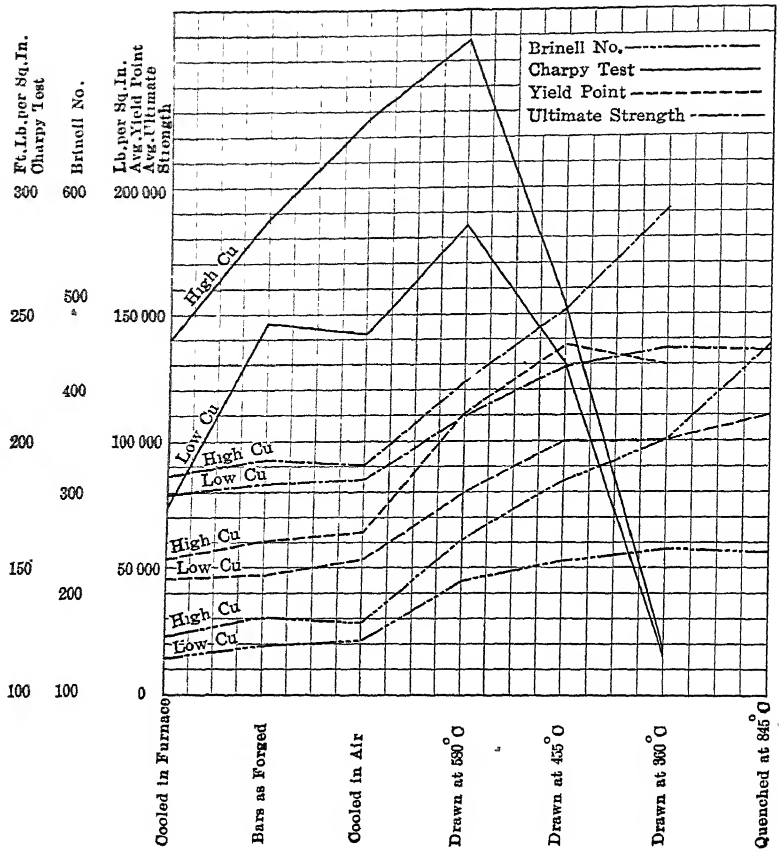


Fig. 1.

and ultimate strength are in every case higher while the ductility is practically the same, although here too the average figures for reduction of area are with one exception slightly higher for the high-copper than for the low. This, however, is offset by slightly higher values for elongation in a majority of the tests in favor of the low-copper.

The hardness tests by both methods show the high-copper steel in all tests to be harder than the low-copper.

The Charpy shock tests show the high-copper steel in all cases to be superior to the low-copper.

In general, the results confirm the work of Ball, Stead, Breuil, and Campbell as regards the effect of copper on hardness and tensile strength. They confirm the work of Breuil as regards brittleness and the work of Campbell as regards reduction of area. It is also true that, as Stead has stated, the behavior of the copper steel resembled that of nickel steel.

## Discussion

Frank N. Speller,* Pittsburgh, Pa.—Mr. Hayward's paper again draws attention to the fact that copper in steel is not the enemy that a good many of us at one time thought it was. He shows clearly that steel with over 0.75 per cent. of copper is as dependable, under the conditions which he describes, as the same steel without copper.

I might add that you can weld copper-steel satisfactorily up to that point; in fact, we have succeeded in welding pipe by the lap weld process up to 1 per cent. copper. Notwithstanding the data that have been presented on the effects of copper on the physical properties and corrosion of steel, a maximum copper limitation is still found in many specifications. In some cases the limit is extremely low, such as below 0.05 or 0.03 per cent., or thereabout. That is a relic of days gone by, which should be discouraged.

E. F. Cone, New York, N. Y.—Not so many years ago steel-casting makers were very particular as to the amount of copper that existed in the pig iron they used. They would not use copper-bearing pig iron unless mixed with non-copper iron. I do not know what the practice now is in the particular foundry in which I was metallurgist at one time, but I do know that a large converter steel casting company at the present time is using copper-bearing pig iron entirely, containing as much as 0.25 or 0.50 per cent. of copper. Its president is firm in the conviction that the physical properties of the castings which he now makes are at least 10 per cent. better than they were when he used non-copper-bearing pig iron, although his metal contains from 0.75 to 1 per cent. copper.

Dr. J. A. Mathews,† Syracuse, N. Y.—I agree with Mr. Speller that the copper limitations should be eliminated from a number of specifications. We have gone beyond the time when it seemed necessary to limit copper to 0.05 per cent. I know that some Government specifications limit it to 0.05 per cent., for what reason, I cannot say. I recall reading a few years ago a paper by Bischoff, giving analyses of some 25 Swedish irons, which carried copper from traces up to 0.2 per cent., and the list included many of the choicest brands of Swedish iron.

H. M. Boylston, Cambridge, Mass.—Professor Clevenger[1] some time ago discussed a paper on a similar subject, and his ideas seemed to agree with those outlined by Professor Hayward; that the steel was more sorbitic with high copper than with low copper, and that there is always a slight tendency to the balling up of the cementite in the form that we know as spheroidized cementite.

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* National Tube Co.

† President and General Manager, Halcomb Steel Co.

[1] Discussion on "The Influence of Copper upon Steel," *Trans.* (1913), **47,** 566.

# INDEX

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. Casual notices, giving but little information, are usually indicated by bracketed page numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page numbers only.]

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